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MINING AND METALLURGY

Bulletin 61

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

ABSTRACT OF CURRENT DECISIONS
ON
MINES AND MINING

OCTOBER, 1912, TO MARCH, 1913

BY

J. W. THOMPSON

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PREFACE.

This bulletin differs somewhat from other published reports of the Bureau of Mines, which relate chiefly to the bureau's researches into the methods of increasing health, safety, economy, and efficiency in the mining, quarrying, metallurgical, and miscellaneous mineral industries of the country.

This publication is devoted exclusively to a review and abstracts of decisions based on the laws governing the rights and duties of mine owners, operators, miners, and persons trafficking in all kinds of mining properties. It includes abstracts of current decisions of all the Federal and State courts of last resort on questions relating to the mining industries.

The bureau proposes to issue similar bulletins with sufficient frequency to keep the report of the courts mentioned reasonably current. There is no present medium known to the bureau by which this information is conveyed immediately to miners, mine owners, or other persons interested in mining and mining industries.

The interest manifested in this bulletin by the persons for whom it is intended will be taken as a basis for determining whether the subsequent publication of similar bulletins is warranted. The issuance of such bulletins will accordingly depend on the number of individual requests received for this one and on the opinion as to its value expressed by those who receive it.

J. A. HOLMES.

ABSTRACT OF CURRENT DECISIONS ON MINES AND MINING, OCTOBER, 1912, TO MARCH, 1913.

By J. W. THOMPSON.

MINERALS AND MINERAL LANDS.

A. SALE AND CONVEYANCE.
B. SURFACE AND MINERALS—OWNERSHIP AND SEVERANCE.

A. SALE AND CONVEYANCE.

CONVEYANCE OF COAL IN PLACE.

Coal or other mineral in place may be granted and conveyed as land, separate and apart from that which underlies or overlies it.

Board, etc., of Greene County *v.* Lattas Creek Coal Co., 100 Northeastern, 561 (Indiana), January, 1913.

MINERALS SUBJECT OF GRANT.

Coal, mineral, and stone under the surface of the earth are the subject of grant and exception, and when excepted in a deed become a separate and distinct inheritance.

Gordon *v.* Million, 154 Southwestern, 99 (Missouri), February, 1913.

CONVEYANCE BY ONE COTENANT.

One cotenant of land can not make a valid conveyance as against his cotenants of the minerals therein and a right to the small timber on the surface necessary for mining purposes with easements and full rights of way over the premises by the construction of roads, tramways, or railroads for the purpose of extracting, storing, handling, and shipping the mineral products.

Ball *v.* Clark, 150 Southwestern, 359 (Kentucky), November, 1912.

CONTRACT OF SALE—COVENANTS RUNNING WITH LAND.

Where a deed was made pursuant to a contract for the sale of certain copper-mining property at an agreed price, and at the same time and as a part of the same transaction a written agreement was

entered into providing for the payment of the balance of the unpaid purchase money out of the net profits resulting from the operation of the mining property, and also providing that it should be binding upon the successors and assigns of the parties, this agreement was not a mere personal contract to be performed by the promissor, but was a covenant running with the land and binding upon a purchaser and of which he was bound to take notice.

Henchman *v.* Consolidated Arizona Smelting Co., 198 Fed., 907, August, 1912.

CONTRACT OF SALE OF MINES—TIME IS OF THE ESSENCE.

Where mines or mining properties are the subject of a contract of sale, time is of the essence, independent of any express stipulation inserted in the contract, and where time is expressly made the essence, then an inexcusable failure to perform or to pay according to the conditions terminates the rights of the promissor under the contract, and if his right of possession depends on his making payments at stated times, his failure to pay accordingly terminates his right to possession and no notice is necessary; and a tender of the amount due after default is not available in the absence of a waiver by the promisee.

Champion Gold Mining Co. *v.* Champion Mines, 128 Pacific, 315 (California), November, 1912.

CONTRACT BY OFFICERS WITH STOCKHOLDER OF ANOTHER CORPORATION.

A verbal agreement entered into between the president and general manager of an oil company with a stockholder of such company who was also the manager and assistant treasurer of another oil company, by which certain mining properties of the first-named company and of the president, who claimed an interest in his own right, were sold to such purchasing company in consideration of a certain sum paid into the treasury of the first-named company, was binding upon both corporations, though made for the benefit of the selling corporation; and the latter had the right to maintain an action to recover the amount so paid from such stockholder and assistant treasurer of the purchasing company and from another stockholder of the selling corporation who subsequently appropriated the amount so paid under the claim that the sale was for their interest in the property.

Love *v.* Kirkbride Drilling & Oil Co., 129 Pacific, 858 (Oklahoma), January, 1913.

DEED CONVEYING COAL MINES.

A deed conveying "the equal undivided half of the coal mines," situated on a certain described tract of land, and reciting that "said mines are now showing themselves in the bed of the creek running

through said described tract of land," is intended to convey the coal deposits subject to being mined and is not limited to the particular veins of coal shown by the croppings in the creek, but cover an undivided half of all the coal deposits underlying the land described.

Gordon *v.* Million, 154 Southwestern, 99 (Missouri), February, 1913.

A purchaser of a mine who fails to make payments according to his purchase contract, and at the stated times, where time is of the essence, forfeits his right to continued possession and also forfeits the payments made, where such forfeiture is expressly provided in the contract.

Champion Gold Mining Co. *v.* Champion Mines, 128 Pacific, 315 (California), November, 1912.

OPTION CONTRACT TO PURCHASE MINES—CONSTRUCTION AND FORFEITURE.

An option contract to purchase mines provided that the purchase price was to be payable in installments, and possession was to be delivered on the first payment, and the purchaser was to expend not less than $100,000 in the development of the mine the first year and to pay the vendor 10 per cent of the gross amount of bullion obtained from clean-ups, such amounts to be applied on the installments of the purchase money, and upon the failure to make any specified payment, or upon breach of any of the agreements, all payments made were to be forfeited, and the vendor could reenter and take possession of the property. Under this contract the purchaser's right to continue in possession and operate the mine depended on the payment of the 10 per cent of the gross amount of each clean-up, and a tender after maturity of an amount not sufficient to pay the percentage on clean-ups admittedly due did not prevent a forfeiture of the contract at the election of the vendor.

Champion Gold Mining Co. *v.* Champion Mines, 128 Pacific, 315 (California), November, 1912.

OPTION TO PURCHASE COAL.

A contract of sale and an agreement to convey a certain vein of coal in and under a certain described tract of land, with the right to mine and remove the same, for a stipulated price payable at certain stated dates, is no more than an option where it provides that it shall be null and void if the wife of the vendor refuses to sign the deed.

Thompson *v.* Craft, 85 Atlantic, 1107 (Pennsylvania), January, 1913.

OPTION CONVERTED INTO ABSOLUTE CONTRACT OF SALE.

An option contract to sell and convey a certain vein of coal in and under a certain described tract of land, with the right to mine and remove the same, providing that it should be null and void if

the vendor's wife should refuse to sign the deed, became absolute where the vendor on his acceptance of an assignment of the contract indorsed thereon that the "contract of sale is made absolute" and acknowledged receipt of part of the purchase price, and the assignee may compel specific performance of the agreement subject to the wife's right of dower, where the wife refused to join in the conveyance.

Thompson *v.* Craft, 85 Atlantic, 1107 (Pennsylvania), January, 1913.

ACCOUNTING FOR PROFITS UNDER OPTIONS.

In an action for an accounting for profits realized from the operation of a certain mining property where it appeared that the complainant and the defendant entered into a verbal agreement by which the defendant should obtain options upon two separate mining properties, and the complainant should advance money sufficient to operate the same by means of leases on a royalty basis, and where the defendant according to the agreement obtained an option on one of the properties and the complainant advanced money according to the agreement, and subsequently the defendant obtained an option for himself solely on the other mining property, and both were operated through leases on a royalty basis, the defendant was bound to account to the complainant for the profits realized by him from the operation of such second property, as the original agreement was entire and included both properties alike.

Quinn *v.* Tully, 140 Northwestern, 492 (Michigan), March, 1913.

AGREEMENT TO RECONVEY—NOTICE.

On the sale of certain mining property the seller made a written agreement that if the purchaser was dissatisfied with the property at the end of the three years the seller would return the amount paid on reconveyance of the property; the purchaser must, in order to hold the vendor, give notice of his dissatisfaction within a reasonable time after the expiration of the three years, and notice of such dissatisfaction, given two and one-half months after the end of the three years, is within a reasonable time where the vendor was in the Territory of Alaska at the expiration of the three years' time and notice could not be served upon him earlier.

McDougall *v.* O'Connell, 130 Pacific, 362 (Washington), February, 1913.

EJECTMENT TO DETERMINE TITLE TO COAL.

Ejectment is the proper remedy to try the title to an estate in coal mines under a deed conveying to the plaintiff the coal mines situated on a certain tract of land described, where the defendant was in the

possession of the surface and had in digging a hole discovered a vein of coal and was claiming the entire estate from the surface to the center of the earth.

Gordon v. Million, 154 Southwestern, 99 (Missouri), February, 1913.

SALE INDUCED BY FRAUD.

When a purchase of mineral lands is induced by the fraudulent representation that gold in great quantities had been found upon ground immediately adjacent to the lands purchased, it is proper to assume that the purchase was induced solely by the representations concerning the gold deposits upon the adjoining lands, and whether the purchaser was told by the seller that gold in paying quantities had been found there or whether the seller presented the purchaser with written logs and copies of assays showing rich gold deposits with assurances that they were correct is immaterial; and it is not even material whether the seller knew that his representations were false, as the test is the truth of the representations and whether the purchaser relied upon them and believed them to be true.

Kleine v. Gidcomb, 152 Southwestern, 462 (Texas), January, 1913.

B. SURFACE AND MINERALS—OWNERSHIP AND SEVERANCE.

SEPARATE OWNERSHIP OF SURFACE AND MINERALS.

The surface ownership of land may be in one person, and that of the minerals therein in another, and both are, in such cases, landholders.

Ball v. Clark, 150 Southwestern, 359 (Kentucky), November, 1912.

ESTATE IN COAL DEPOSITS.

Coal deposits when separated by grant or reservation in a deed are as much of an estate in lands as is the surface of the same lands.

Gordon v. Million, 154 Southwestern, 99 (Missouri), February, 1913.

PARTITION OF MINERALS.

An estate in minerals is a subject of partition.

Ball v. Clark, 150 Southwestern, 359 (Kentucky), November, 1912.

SEVERANCE OF SURFACE AND MINERALS.

The owner of land may by proper conveyance to another sever the minerals from the surface and each will thereafter constitute a separate and distinct estate, taxable as such.

McCormick v. Berkley, 86 Atlantic, 97 (Pennsylvania), January, 1913.

There may be a complete severance of the mineral estate in lands from the surface estate.

Gordon v. Million, 154 Southwestern, 99 (Missouri), February, 1913.

SEPARATE CONVEYANCE OF SURFACE AND MINERALS.

The owner of land may by deed containing a sufficient description convey all the surface to one person, reserving the mineral therein, and may by a proper deed containing sufficient description convey the mineral in the same land to another person.

Bingham *v.* Carnes, 153 Southwestern, 193 (Kentucky), February, 1913.

The owner of land may convey the surface to another and reserve to himself an estate in fee in the minerals, or he may convey the surface to one person and the minerals to another, and such a conveyance will create an estate separate and distinct in the properties sundered the one from the other, entire and complete in fee simple.

Ball *v.* Clark, 150 Southwestern, 359 (Kentucky), November, 1912.

GRANT OF COAL—USE OF SURFACE.

The grant of coal in land carries with it the use of the surface so far as is necessary to carry on mining operations.

Gordon *v.* Million, 154 Southwestern, 99 (Missouri), February, 1913.

OIL AND GAS LANDS—OWNERSHIP OF OIL.

Oil in and under the surface of the earth and not reduced to actual possession constitutes a part of the land and belongs to the owner of the land, and he has the right to reduce it to possession or grant the privilege of doing so to other persons.

Kahle *v.* Crown Oil Co., 100 Northeastern, 681 (Indiana), January, 1913.

AUTHORITY OF INDIANS OVER OIL AND GAS—POWER TO SELL.

Under the act of Congress dated June 28, 1906 (34 Stat., 540), the adult members of the Osage Tribe of Indians are not authorized to sell the oil, gas, coal, or other mineral covered by their lands, though certificates of competency were issued to such Indians by the Secretary of the Interior.

Neilson *v.* Alberty, 129 Pacific, 847 (Oklahoma), January, 1913.

MINING CLAIMS.

A. Location and Effect.
B. Assessment Work.
C. Abandonment and Relocation.
D. Adverse Claims.
E. Liens.
F. Coal Locations.

A. LOCATION AND EFFECT.

DISCOVERY NECESSARY.

A valid location of a placer-mining claim can not be made without a discovery of mineral, nor can such location be made in the State of Washington without complying with the statute by commencing excavations or the construction of works within the required time. The labor required by the statute must be performed.

Spokane Portland Cement Co. *v.* Larson, 128 Pacific, 641 (Washington), December, 1912.

There can be no valid location of a mining claim until a sufficient actual discovery of mineral is made on the claim and no question of the doing of annual assessment work is involved, as it is only after such a discovery, when actual possession is no longer necessary to protect the location against subsequent locators, that annual assessment work is essential to prevent a forfeiture.

Borgwardt *v.* McKittrick Oil Co., 130 Pacific, 417 (California), March, 1913.

DISCOVERY OF MINERAL.

In a contest involving the right of possession of a mining claim it is competent for expert witnesses to give their opinion as to whether or not there is such a showing of a vein and mineral in the location as would justify a reasonably prudent miner in the expenditure of his money and time, with the expectation of finding minerals in profitable quantities.

King Solomon Tunnel & Development Co. *v.* Mary Verna Mining Co., 127 Pacific, 129 (Colorado), October, 1912.

CHARACTER OF MINERALS SUBJECT TO LOCATION.

In a contest between a placer and a lode locator a court has jurisdiction to determine whether the mineral land in controversy was of a character which entitled it to be located as a placer mine, or whether it could only be entered as a lode mining claim, for the reason that if the land was not subject to location as a placer claim the placer claimant obtained no possessory right thereto.

San Francisco Chemical Co. *v.* Duffield, 201 Fed., 830, p. 834, November, 1912.

GOOD FAITH LOCATION.

In an action of ejectment the question of the good faith of a locator is for the jury to decide, as a court can not lay down rules by which to distinguish a speculative location from one made in good faith and with the intent to make excavations and ascertain the character of

the vein so as to determine whether it will justify the expenditures required to extract the metal.

Rooney *v.* Barnette, 200 Fed., 700, p. 709, October, 1912.

LANDS ALREADY APPROPRIATED.

No right to a mining claim can be initiated while the ground is in possession of another who has the right to its possession under an earlier lawful location, nor can such a claim be initiated by forcible or fraudulent entry upon land in possession of one who has no right either to the possession or to the title.

San Francisco Chemical Co. *v.* Duffield, 201 Fed., 830, p. 834.

PRIORITY OVER CONFLICT GROUND.

Where, prior to the patent survey of a second mining claim, the south end line of an existing and prior claim was moved to the south of the line originally monumented, to correspond with the call in the location notice and certificate, prior to the patent survey of such second claim, and where subsequently the easterly side line of such second claim was moved to the east, the owner of such original claim has a better right than the owner of such second claim to the ground in conflict occasioned by such changes in the boundaries of the respective claims.

Indiana Nevada Mining Co. *v.* Gold Hill Mining & Milling Co., 126 Pacific, 965 (Nevada), October, 1912.

MARKING BOUNDARIES.

The marking upon the ground of the boundaries of a mining location should be made so certain and so plain that anyone prospecting in the same locality would have no trouble in locating the exact ground claimed; but any markings on the ground by stakes, monuments, mounds, and written notices whereby the boundaries can be readily traced, are sufficient, and if a third party intending to locate can readily ascertain from what has been done by the prior locator the extent and boundaries of the existing location, then the statute has been sufficiently complied with.

Madeira *v.* Sonoma Magnesite Co., 130 Pacific, 175 (California), December, 1912.

FAILURE TO MARK BOUNDARIES.

A locator of a mining ground who failed to definitely mark the boundaries of his location and left his claim unmarked from September, 1905, to June, 1906, and knew that another person had located a part of the same ground and performed a large amount of labor

thereon without knowledge of such prior location, did not act within a reasonable time in definitely marking his claim on the ground so that its boundaries could be readily traced.

Madeira *v.* Sonoma Magnesite Co., 130 Pacific, 175 (California), December, 1912.

EXCESSIVE GROUND.

A mining location duly made according to the mining laws is not invalid where the locator includes within the boundaries of his claim more than the law permits, as in such case he is entitled to hold to the limit which the law authorizes within the limits he has laid out, and the territory embraced within his boundaries in excess of such limits is to be rejected; but where a locator relies upon the corners established and marked out, a different rule governs, and if the courses are so widely separated from where they should be as to bear no relation to the lode, and so remote as to justify reasonable inference that they were not intended to apply to the lode in question, they would add little if any force to the claim that the law had been complied with, and especially so where the notice once posted at the discovery point had disappeared or where the lode line was not distinctly marked.

Madeira *v.* Sonoma Magnesite Co., 130 Pacific, 175 (California), December, 1912.

ENTRY ON EXISTING LOCATION.

A provisional location can not be made upon an existing mining claim with the intention on the part of the locator that the validity of such provisional location depends on whether or not the prior locator fails to do the annual assessment work and thereby forfeits the existing claim.

Rooney *v.* Barnette, 200 Fed., 700, p. 708, October, 1912.

PLACER CLAIMS.

By a placer claim is meant ground within defined boundaries containing mineral in its earth, sand, or gravel—ground that includes valuable deposits not in place, nor fixed in rock, but which are in a loose state and that may usually be collected by washing or amalgamation without milling.

San Francisco Chemical Co. *v.* Duffield, 201 Fed., 830, p. 836, November, 1912.

PLACER AND LODE CLAIMS—ROCK PHOSPHATE.

Calcium phosphate or rock phosphate found in place having a dip and strike, firmly fixed in the mass of a mountain and occurring between strata of limestone, chert, and shale, where the line of demarca-

tion between veins of such phosphate rock and wall rock of limestone or shale is well defined and distinct, and where the distinction between such phosphate rock, having commercial value, and the wall rock, with no commercial value, is readily determined by visual inspection, is subject to location under the mining laws only as a vein or lode and not as a placer claim.

San Francisco Chemical Co. *v.* Duffield, 201 Fed., 830, p. 836, November, 1912.

PAROL AGREEMENT TO LOCATE FOR BENEFIT OF OTHERS.

An agreement to locate a mining claim for the benefit of others is valid though not in writing.

Clark *v.* Mitchell, 130 Pacific, 760 (Nevada), March, 1913.

AGREEMENT TO LOCATE FOR ANOTHER—ESTOPPEL.

An agent or a representative of an owner of a mining claim who agreed to perform the assessment work for a certain year for an undivided interest in the claim and to relocate another claim for such owner in consideration of an undivided interest therein is, in performing such work, and relocating such other claim, bound to protect the original claimant to the extent of the interest agreed upon and is estopped from denying the rights of such original claimant and can not question the validity of the locations made by himself in violation of the agreement.

Clark *v.* Mitchell, 130 Pacific, 760 (Nevada), March, 1913.

LOCATIONS FOR BENEFIT OF LOCATORS.

Valid locations of mining claims may be made by a number of different locators, though made under an arrangement by which each locator was to transfer his claim to a corporation to be subsequently organized and in which the locators should be the sole stockholders, each owning an equal undivided part of the stock.

Borgwardt *v.* McKittrick Oil Co., 130 Pacific, 417 (California), March, 1913.

POSSESSORY RIGHT.

A locator of a mining claim has no vested right which courts are obliged to recognize until the inchoate location is perfected by discovery, but where such location is made in good faith the locator has the right as against third persons to be protected against all forms of forcible, fraudulent, surreptitious, or clandestine entries so long as he remains in possession and with due diligence prosecutes his work toward discovery.

Borgwardt *v.* McKittrick Oil Co., 130 Pacific, 417 (California), March, 1913.

POSSESSION PROTECTED.

The possession of an attempted locator is protected only while he may fairly be held to be actually engaged in such work as may reasonably be held to be discovery work.

Borgwardt v. McKittrick Oil Co., 130 Pacific, 417 (California), March, 1913.

POSSESSORY RIGHTS AND RELOCATION.

The location of mineral ground gives to the locator before discovery, and so long as he complies with the United States statutes and State laws and local rules and regulations, the absolute right of possession as against all intruders, and he may convey or transfer this right to another; and during such time such ground is so segregated it is not open to location by another, and any attempted relocation of such ground during such time is void.

Rooney v. Barnette, 200 Fed., 700, p. 709, October, 1912.

AMENDED LOCATION.

The filing of amended location certificates for the purpose of correcting errors in the original location, without waiving previously acquired rights, is not a waiver of any priority obtained by the original location.

King Solomon Tunnel & Development Co. v. Mary Verna Milling Co., 127 Pacific, 129 (Colorado), October, 1912.

FAILURE TO FILE AMENDED CERTIFICATE.

The failure to file a certificate or amended certificate of location, as required by the statute of Nevada, has the effect of shifting the burden of proof, but does not otherwise affect the rights of the parties thereunder.

Indiana Nevada Mining Co. v. Gold Hill Mining & Milling Co., 126 Pacific, 965 (Nevada), October, 1912.

EFFECT OF PATENT.

A patent for a mining claim that contains within its surface boundaries the location and monument shaft of another claim effectually extinguishes the latter claim as a valid location.

Indiana Nevada Mining Co. v. Gold Hill Mining & Milling Co., 126 Pacific, 965 (Nevada), October, 1912.

LOCATION OF HIGHWAY.

A board of commissioners of a county has the power to locate and establish a public highway over an existing mining claim, as this amounts to no more than establishing an easement over such claim to the extent necessary for use as a highway.

Olaine v. McGraw, 120 Pacific, 460 (California), January, 1913.

B. ASSESSMENT WORK.

ASSOCIATION CLAIM.

The mining laws do not require the annual assessment work on each 20-acre tract of an association claim.

Rooney *v.* Barnette, 200 Fed., 700, p. 708, October, 1912.

PAROL AGREEMENT TO PERFORM ASSESSMENT WORK AND RELOCATE CLAIM.

An oral agreement by which one person agreed to perform the assessment work upon a certain mining claim for a year, in consideration of which the owner of the claim agreed to convey the undivided one-fourth interest of such claim, is not void under the statute of frauds where the owner of the claim fully performed his part of the agreement; but where the other party with the fraudulent purpose of dispossessing the owner of his claim willfully failed to perform such assessment work, causing the claim to be forfeited and to revert to the public domain and afterwards relocated such claim in the names of himself and other persons, a trust relation was thereby created and the original owner can enforce the trust and recover the share of such relocated mine which he would have received had the other party performed his part of the contract.

Clark *v.* Mitchell, 130 Pacific, 760 (Nevada), March, 1913.

INSUFFICIENT PROOF.

It is not sufficient proof of assessment work for a party to state in his affidavit that he has in his possession the affidavits of three competent persons to the effect that the assessment work on a mining claim for a particular year has been done.

Anderson *v.* Robinson, 127 Pacific, 546 (Oregon), November, 1912.

VALUE OF HYDRAULIC WORK.

The value of assessment work by hydraulicking on a mining claim is not determined by the wages of the man who holds the nozzle, but by the results accomplished, and in such case the value of the use of the plant, including the water rights, ditches, pipe lines, giants, all of which contribute more to the result than the manual labor, must be considered.

Anderson *v.* Robinson, 126 Pacific, 988 (Oregon), October, 1912.

CONSTRUCTION OF FLUME FOR OPERATION OF MINE.

The construction of a flume for the bringing of water to a mine for the sole purpose of working it, where this is the only way by which the mine could be worked or prospected, is development work within the

meaning of the California statute giving a laborer lien for labor performed in developing a mine.

McClung *v.* Paradise Gold Mining Co., 129 Pacific, 774, p. 776 (California), January, 1913.

RESUMPTION OF WORK.

It is immaterial whether a locator or owner of a mining claim performed the assessment work for a particular year if such locator or owner was subsequently in possession and performing work on the claim; and it is not then open to relocation and possession by another, and work thereafter performed would apply on the work required for that year.

Anderson *v.* Robinson, 126 Pacific, 988 (Oregon), October, 1912.

C. ABANDONMENT AND RELOCATION.

QUESTION OF FACT.

The abandonment of a mining claim by a locator is a question of fact, and the fact is to be found from the intention and the filing of amended location certificate; but the sinking of new discovery shafts are not necessarily evidence of an intention to abandon where the amended location certificate expressly stated that it was without waiver of previously acquired rights.

King Solomon Tunnel & Development Co. *v.* Mary Verna Mining Co., 127 Pacific, 129 (Colorado), October, 1912.
Indiana Nevada Mining Co. *v.* Gold Hill Mining & Milling Co., 126 Pacific, 965 (Nevada), October, 1912.

PRIORITY OF RIGHT.

In an action to recover possession of a mining claim where the defendant pleaded title to the disputed ground by reason of discovery and location antedating that of the plaintiff's, the fact that the defendant did, subsequent to the location claimed by the plaintiff, file amended locations does not of itself show an abandonment where the amended location notices stated that they were amended locations, made to correct errors in the original locations and without waiver of previously acquired rights; and the further fact that the defendant sunk new discovery shafts under such amended location did not break the continuity of the original location where it clearly appeared that the locator did not intend to abandon the original location.

King Solomon Tunnel & Development Co. *v.* Mary Verna Mining Co., 127 Pacific, 129 (Colorado), October, 1912.

ABANDONMENT OF PLACER CLAIM.

An attempt to locate a placer mining claim and appropriate alleged water rights as a speculative venture in an effort to prevent others from appropriating, developing, or using the waters of a lake or stream

will not operate as a valid location where the locator and his successors made no honest effort to work the claim for more than a year, and where they made no application to the Forest Service of the United States for permission to develop or work the claim; and under such circumstances an abandonment may properly be declared.

Spokane Portland Cement Co. *v.* Larson, 128 Pacific, 641 (Washington), December, 1912.

NEW DISCOVERY UNDER AMENDED LOCATION.

Under the statute of Colorado, the locator of a mining claim who files an amended location certificate for the purpose of correcting errors in the original location is not required to discover a new vein or lode, or sink any additional shaft, or make any new discoveries of mineral on his location.

King Solomon Tunnel & Development Co. *v.* Mary Verna Mining Co., 127 Pacific, 129 (Colorado), October, 1912.

D. ADVERSE CLAIMS.

EVIDENCE AS TO VACANT PUBLIC LAND.

In an action on adverse claims involving the priority of locations of mining claims witnesses may not testify as to whether or not the located land was vacant public domain at the time of a certain location, but in such case the witnesses are limited in their testimony to their knowledge of the physical conditions of the ground, based on personal examination and inspection thereof, and as to what shafts, adits, tunnels, or other mining developments are located thereon, and it is then the province of the court and jury to determine whether or not the alleged locations were made upon unappropriated public domain.

King Solomon Tunnel & Development Co. *v.* Mary Verna Mining Co., 127 Pacific, 129 (Colorado), October, 1912.

CONFLICT IN LODE AND PLACER LOCATIONS.

In an action on an adverse claim to determine a controversy between lode and placer locators, the court can only inquire as to the priority of location and whether all the requirements of the statute have been complied with.

San Francisco Chemical Co. *v.* Duffield, 201 Fed., 830, p. 833, November, 1912.

TRESPASS AND DAMAGES.

The locator of a lode claim can not successfully maintain an action to enjoin a trespass and to recover damages for such trespass upon his alleged mining location where the testimony shows that such

lode location, triangular in form, overlapped a placer location, and where no discovery was made within the limits of such lode location, but where a surface discovery only was claimed, and where the ore was taken by the defendant from such overlap and wholly within the boundaries of the placer location.

Olaine v. McGraw, 120 Pacific, 460 (California), January, 1912.

E. LIENS.

LIEN FOR LABOR.

In asserting a lien for labor on a mining claim under the statute of California it is not necessary that the claimant should use the language of the statute, nor does the statute require the claimant to state the character of his labor nor show that the labor was performed in development or work by the subtractive process, though he must show by his proof that it was of such kind as is made lienable by the statute and that it did constitute development work or mining by the subtractive process.

McClung v. Paradise Gold Mining Co., 129 Pacific, 774 (California), January, 1913.

NATURE OF WORK AND AUTHORITY OF AGENT.

A person having in contemplation the purchase of a large block of the shares of a corporation owning mining property, and who took an option from the corporation for that purpose and by which he was authorized to go upon its property and repair the flume as he thought necessary and sufficient to run water, and to deliver to its treasurer one-half of all the gold taken from the mine, was sufficient to constitute such officer the statutory agent for the employment of laborers, and to put the corporate officers on notice that he intended to operate the mine and to engage laborers who would be entitled to a lien for their services.

McClung v. Paradise Gold Mining Co., 129 Pacific, 774, p. 776 (California), January, 1913.

PRIORITY OF LIENS.

An agent employed by an English corporation to superintend its mining operations in certain States is entitled to a lien on the mining property for advances made by him and used in the development of the property of the company, and in purchasing certain mining claims for the company to which he took title to himself as security; and such liens are not subject to a floating lien created by an English debenture that does not restrict or in any way interfere with the corporation from prosecuting the business for which it

was organized in foreign countries and there encumbering, transferring, or otherwise dealing with its property in the ordinary course of business.

Pearson *v.* Harris, 200 Fed., 10, October, 1912.

LIEN ON OIL AND GAS LEASEHOLD.

The statute of Kansas giving a lien on the whole of any leasehold for oil or gas purposes, or any oil or gas pipe lines, for labor performed, or for material furnished, or for machinery and oil-well supplies used in digging, drilling, torpedoing, completing, operating, or repairing any oil or gas well, does not give a person performing such labor or furnishing any such materials a lien upon the oil produced.

Black *v.* Gearth, 128 Pacific, 183 (Kansas), December, 1912.

MECHANIC'S LIEN ON OIL WELL.

A leasehold estate, for the purpose of producing oil and gas, and an oil and gas well and its appliances located thereon, are subject to a mechanic's lien, and such oil well is a "structure" within the meaning of the statute giving a mechanic's lien on buildings and structures.

Kanawha Oil & Gas Co. *v.* Wenner, 76 Southeastern, 893 (West Virginia), December, 1912.

LABOR LIEN—WAIVER.

An agreement by which a mining company leased to a person who had performed work for it certain oil wells for a specified time upon a royalty of 10 cents per barrel for all oil produced, and by which the lessee was to make certain collections on account of oil previously sold by the company to third persons, and apply the proceeds of such collections to his claim for such work performed by him, and also agreeing to sell oil in certain holes and apply the proceeds upon such indebtedness, did not constitute a novation so as to extinguish the existing indebtedness and did not operate as a waiver of the lien held by the lessee upon the mining property of the lessor for the services rendered.

Reynolds *v.* York Syndicate Oil Co., 130 Pacific, 183 (California), December, 1912.

ASSIGNMENT OF LABOR LIEN.

A laborer's lien on a mine under the statute of California is not invalid merely because the date of the assignment antedated that of the recording, where the assignment was made to take effect after the recording of the claim as a lien and where the assignment itself showed that it was for the claim of lien "heretofore filed" by the assignor against the property of the mining company.

McClung *v.* Paradise Gold Mining Co., 129 Pacific, 774, p. 777 (California), January, 1913.

F. COAL LOCATIONS.

ENTRY FOR SOLE USE OF ENTRYMAN.

The officers of the land office have the right to assume that an application to purchase coal lands is made with knowledge of the law regulating such purchases and that the applicant is applying to purchase the land for his own use and benefit, as the law restricts the amount of land that can be purchased by one person to 160 acres and by an association to 320 acres.

United States v. Home Coal & Coke Co., 200 Fed., 910, p. 914, October, 1912.

ENTRY BY ONE PERSON FOR BENEFIT OF OTHERS.

There is no prohibition, either express or implied, in the coal-land law against an entry by a qualified person for the benefit of another person or association where such person or association is qualified to make the entry in his or its own name and is not seeking to evade restriction in respect of quantity; but the prohibition is against a qualified person making an entry for another who is disqualified.

United States v. Home Coal & Coke Co., 200 Fed., 910, p. 917, October, 1912.

STATEMENT AS TO PURPOSE OF ENTRY.

The statute regulating the entry and purchase of coal lands does not require an applicant to state in his application that the entry is for his sole use and benefit.

United States v. Home Coal & Coke Co., 200 Fed., 910, p. 914, October, 1912.

PURCHASE MONEY REFUNDED.

Where from any cause a coal entry has been erroneously allowed the purchase money may be repaid to the entrymen; but the expression "erroneously allowed" denotes some mistake or error on the part of the land officers whereby an entry is allowed when it should be disallowed, and not some fraud or false pretense practiced on them whereby an applicant appears to be entitled to the allowance of an entry when in truth he is not.

United States v. Home Coal & Coke Co., 200 Fed., 910, p. 916, October, 1912.

MINES AND MINING OPERATIONS.

A. DUTIES AND LIABILITIES OF OPERATOR.
B. RIGHTS AND DUTIES OF MINER.
C. CONTRACTS RELATING TO OPERATIONS.
D. METHODS OF OPERATING—RIGHTS AND LIABILITIES.
E. STATUTORY REGULATIONS—EFFECT AND NONOBSERVANCE.
F. NEGLIGENCE OF OPERATOR—LIABILITY.
G. CONTRIBUTORY NEGLIGENCE OF MINER.
H. ASSUMPTION OF RISK BY MINER.
1. RISKS ASSUMED.
2. RISKS NOT ASSUMED.
I. PROMISE OF OPERATOR TO REPAIR—EFFECT.

A. DUTIES AND LIABILITIES OF OPERATOR.

SAFE WORKING PLACE.

A mine owner owes the duty to the miners to make its mine as reasonably safe to work in as is practicable in such a dangerous vocation.

Cook *v.* Cranberry Furnace Co., 76 Southeastern, 473 (North Carolina), November, 1912.

DANGEROUS EMPLOYMENT.

A coal-mine operator is not an insurer of the safety of its miners and owes a miner no greater duty for his protection than he owes to himself.

Stony Fork Coal Co. *v.* Lingar, 153 Southwestern, 6 (Kentucky), February, 1913.

DUTY TO KEEP WORKING PLACE SAFE.

The duty of a mine operator to keep safe the place where a miner works extends to places immediately connected with the working place of the miner where he must necessarily go in connection with his ordinary work as a miner, and a miner did not forfeit his right to the protection the discharge of the operator's duty would afford where he was temporarily in an unprotected part of his chamber, without notice of danger, to get a needed drink from his water pail, which he kept in close proximity to his place of work.

Mammoth Mining Co. *v.* Thomas, 201 Fed., 297, p. 300, October, 1912.

DUTY TO MAKE MINE SAFE.

A mine owner fails to exercise ordinary care to make its mine reasonably safe for its miners where it had knowledge that the roof was in an unsafe and dangerous condition and where slate had been

falling from a roof in an entry for some time and it took no steps to remove the defect either by taking down the loose slate or by propping the roof.

Trosper Coal Co. *v.* Crawford, 153 Southwestern, 211 (Kentucky), February, 1913.

SAFE WORKING PLACE FOR MINERS.

It is the duty of a mine operator to inspect and keep safe a chamber or chute where miners are expected to work up to the line of such chamber or chute, and it is the continuing duty of the operator to warn miners of the danger of such adjacent or adjoining chute, and either protect them or give them opportunity to protect themselves, where the miners themselves are not permitted to enter or inspect such adjoining chute.

Gennaux *v.* Northwestern Improvement Co., 130 Pacific, 495 (Washington), February, 1913.

DUTY TO PROVIDE SAFE PLACE AND WARN MINERS.

A mining company owes the duty to its miners of using ordinary care and diligence to provide reasonably safe places in which miners are to work, and to see that such places are reasonably free from danger, and that the appliances furnished are reasonably safe and free from danger; and it is obligatory upon such company to warn its servants of the situation, so that they may not accidentally or purposely come in contact with the danger, and such warning must be plainly noticeable to give the servant a comprehension of the danger.

Williams *v.* Bunker Hill & Sullivan Mining, etc., Co., 200 Fed., 211, October, 1912.

KEEPING MINE ROOF SAFE.

The rule that a mine owner or operator is required to furnish a safe working place in which the miners may work applies also to places in the mine other than those in which miners work, and a mine owner may be liable for an injury to a miner caused by falling slate or rock from the roof where the miner under direction of the operator or foreman was passing through the mine and injured by the fall of such material from the roof.

Broadway Coal Mining Co. *v.* Robinson, 150 Southwestern, 1000 (Kentucky), November, 1912.

DUTY TO KEEP ROOF OF CROSSCUTS SAFE.

Where a mine was worked by what is called a "panel" system and the entries were connected by crosscuts in which miners were not required to work, but in which, with the knowledge and acqui-

escence of the mine owner, the miners were accustomed to store tools and to sit while eating their dinners or for other convenient uses, it then became the duty of the mine operator to keep the walls and roofs of these crosscuts in reasonably safe condition and make reasonably frequent and proper inspection thereof, and the failure or neglect on his part would render him liable for an injury to a miner while eating his dinner caused by the falling of the roof.

Kennis *v.* Ogden, 138 Northwestern, 467 (Iowa), November, 1912.

DUTY TO FURNISH PASSAGEWAY.

A mining company failing to provide a passageway by which miners and employees can pass in safety from one part of the mine to another without going through the elevator shaft where elevators are constantly being raised and lowered without warning or notice is not exercising ordinary care to furnish its miners a reasonably safe place in which to work, and the fact that no statute of the State required mine owners to provide a passageway around the elevator shaft did not relieve it from liability for injury to a miner while passing through the shaft.

Fluehart Collieries Co. *v.* Elam, 151 Southwestern, 34 (Kentucky), December, 1912.

DUTY TO FURNISH SAFE HAULAGEWAYS.

A mine owner and operator must furnish a safe passageway for drivers of cars hauling coal out of the mine, and it is liable for an injury to a driver caused by protruding rock extending from the rib of the entry to a point perpendicular above the rail, leaving a clearness above the edge of the car of only 2 inches.

Ek *v.* Phillips Fuel Co., 138 Northwestern, 547 (Iowa), November, 1912.

DUTY EXTENDS TO ALL PARTS OF MINE.

The duty of a coal-mine operator to furnish a miner with a reasonably safe place is not confined to the precise spot in which he works, but includes the places to and from which he is required to go in performing his duties, and the places whither he is ordered by the mine foreman to go.

Fluehart Collieries Co. *v.* Elam, 151 Southwestern, 34 (Kentucky), December, 1912.

DUTY TO PROVIDE SUITABLE APPLIANCES.

It is the duty of the master to use due diligence to provide suitable appliances in the operation of his business, and the servant may assume that the master has performed this duty, and an employee does not assume the risk of his employer's negligence in performing such duties, and he is not obliged to pass judgment upon the em-

ployer's methods of transacting his business, but may assume that reasonable care has been used in furnishing the appliances necessary for its operation; but if the conditions are constant and of long standing, and the danger is so obvious as to suggest itself to persons in possession of their faculties and of ordinary intelligence, he can not escape the consequences by refusing to see the conditions or appreciate the danger.

Killeen *v.* Barnes-King Development Co., 127 Pacific, 89 (Montana), October, 1912.

A coal company operating a coal mine was held liable to a rope driver for an injury received by him by reason of its failure to exercise ordinary care to keep its tracks in a reasonably safe condition, and by reason of which the cable used to draw the coal cars caught on the end of a tie and suddenly came loose and struck and injured the rope driver.

Stony Fork Coal Co. *v.* Lingar, 153 Southwestern, 6 (Kentucky), February, 1913.

It is the duty of a coal company operating a coal mine to exercise ordinary care to provide miners and employees with a reasonably safe place in which to work considering the nature of the employment and must use ordinary care to put and keep its tracks in a reasonably safe condition, and if it failed to do this and the rope driver was injured by reason of a wire cable used to draw the loaded cars the company is liable.

Stony Fork Coal Co. *v.* Lingar, 153 Southwestern, 6 (Kentucky), February, 1913.

DUTY TO PROVIDE SAFE PLACE AND APPLIANCES.

It is the duty of an operator of a coal mine to use ordinary care to furnish the employees and miners a reasonably safe place to work and to use ordinary care to furnish such appliances as are reasonably safe under such strains as might be reasonably anticipated in the uses for which they are intended.

Interstate Coal Co. *v.* Shelton, 153 Southwestern, 1 (Kentucky), February, 1913.
Trosper Coal Co. *v.* Crawford, 153 Southwestern, 211 (Kentucky), February, 1913.

DUTY OF MINE OPERATOR TO INSPECT AND GUARD AGAINST DANGER.

It is the duty of a coal company operating a coal mine to use ordinary care in having the roof inspected by its mine foreman where such roof is supported only by pillars, and if the mine foreman knows, or by the exercise of ordinary care should have known, that the mining of coal by miners would affect the roof of an entry, then it was the duty of the company to guard against the danger by more securely propping the roof and making it reasonably safe, and if the place was dangerous and unsafe and if this was known to the mine foreman, or should have been known to him by the exercise of ordi-

nary care, and a miner working in an entry did not know of the dangerous condition of the roof, and by the exercise of ordinary care could not have known of the danger, then the company is liable to such miner for an injury caused by the falling of the roof.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

DUTY TO WARN AND INSTRUCT MINER.

Though the place where a miner works is reasonably safe, yet the master is not thereby relieved from the duty of warning and instructing a young and inexperienced miner that the service is dangerous and is one in which an inexperienced miner would not appreciate or realize the danger.

Bartley *v.* Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1912.

DUTY TO INSTRUCT INEXPERIENCED MINER.

The fact that a person 23 years of age was employed and paid miner's wages while working in a mine, and the further fact that such miner had worked in another coal mine and had, previous to his employment as a miner, been employed as a car driver, will not relieve the mine owner and operator employing him to work in the mine, with knowledge of his inexperience, from the duty of warning him when he changed his occupation from car driving to mining.

Carney Coal Co. *v.* Benedict, 129 Pacific, 1024 (Wyoming), February, 1913.

DUTY TO EMPLOYEE OF INDEPENDENT CONTRACTOR.

A mine owner who had contracted with a third party to mine ore for it in its mine but who operated for itself the cars or skips for hauling out the coal and transporting the employees of such contractor, owed to an employee of such contractor the duty of providing safe passageways and means of conveyance for use in coming from the place of work.

Republic Iron & Steel Co. *v.* Fuller, 60 Southern, 475 (Alabama), December, 1912.

DUTY TO CONVICT MINER.

A coal-mine operator owes a convict working in a mine the duty of doing him no willful harm and of exercising reasonable care for his personal safety, and the doctrine of assumption of risk from the negligence of fellow servants does not apply to such convict, as he is under involuntary servitude and has no power to quit the employment.

Sloss-Sheffield Steel & Iron Co. *v.* Weir, 60 Southern, 851 (Alabama), January, 1913.

DUTY TO GUARD ELECTRIC WIRES.

It is the duty of a mining company to protect its live trolley wires by trough or other practical device, and thus make the place safe for miners who may be working under or near such wire, and especially where the floor of a mine or of a stope is wet and slippery and where there is danger of a miner accidentally slipping and the tools or implements with which he is working thereby coming in contact with such electric wire.

Williams *v.* Bunker Hill & Sullivan Mining, etc., Co., 200 Fed., 211, October, 1912.

DUTY TO FURNISH SAFE ELECTRICAL APPLIANCES.

A mining company operating its tram cars in its mine by electrical power is not bound to use the greatest possible care, but it is its duty to use ordinary care to make such appliances reasonably safe, and the duty of diligence is greater where the mechanism or the agency employed is highly hazardous.

Meola *v.* Quincy Mining Co., 140 Northwestern, 460 (Michigan), March, 1913.

PLACE MADE UNSAFE BY MINER.

A coal company operating a coal mine was held liable to a miner injured by falling of slate from the roof where the mine foreman directed the miner and his partner or "buddy" to mine out a pillar and directed what to do and how to do it, and where after cutting into the pillar 6 or 8 inches and removing the coal the roof fell, causing the injury, and where it was shown that the mine foreman knew that the roof of an entry becomes rotten from the action of the air upon it and admitted that the slate would not have fallen if it had not been loose, but he made no inspection of the roof, though he could have discovered its condition by sounding it with a pick, and where it also appeared that the slate would not have fallen if the 6 or 8 inches of coal had not been removed, but where the condition of the roof was not known to the miner and it was shown that the mine foreman had previously put up one prop.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

PROOF OF INSUFFICIENT INSPECTION.

In an action by a miner for damages for injuries caused by the falling of a part of a roof in a mine the evidence ordinarily must be confined to the condition of the roof in the particular part of the mine where the accident occurred; but where the mine owner introduced evidence to show that the mine was inspected daily

by three competent persons the injured miner then had the right to show that such inspections were neither competent nor thorough, and one way of doing this was to show the faulty and defective condition of the roof of the mine in parts other than where the injury occurred.

Trosper Coal Co. *v.* Crawford, 153 Southwestern, 211 (Kentucky), February, 1913.

DEFECTIVE TRACKS USED FOR HAULING COAL CARS.

It is the duty of a coal-mine operator to use ordinary care to furnish its employees and miners a reasonably safe place in which to work, and if an operator failed to use reasonable care to make the tracks or tram road used for hauling coal from its mine in a reasonably safe condition, and its defective condition was known or could have been known to the operator by the exercise of ordinary care, and was not known to a rope driver, or such defect was not so obvious that the driver could by the exercise of ordinary care in the discharge of his duty have known such defect, and by reason of such defect the car on which the driver was riding was derailed and wrecked and the driver, while using ordinary care for his own safety, was thereby injured, he is entitled to recover damages for such injury.

Bell-Knox Coal Co. *v.* Gregory, 153 Southwestern, 465 (Kentucky), February, 1913.

ACTS OF VICE PRINCIPAL.

Where miners were using a dump car to haul the excavated earth from a prospect shaft in process of sinking, and the employer had placed, or caused to be placed, a block or bumper upon the rail to prevent the car from running over the collar of the shaft, the employer was in duty bound to place a bumper reasonably sufficient for such purpose; and this duty so plainly belonged to the employer in providing a safe place for the miners to work that it was wholly immaterial, in an action by a miner for injuries caused by reason of the bumper being insufficient, as to the person actually performing the mechanical labor in placing the bumper, if the injured miner himself had no part in placing it, and if the bumper was placed by another miner or employee he was, as to the particular act, a vice principal.

Killeen *v.* Barnes-King Development Co., 127 Pacific, 89 (Montana), October, 1912.

CONVICT MINER AS SUPERINTENDENT—LIABILITY.

A coal-mine operator employing convicts to work in its mine may intrust superintendence and authority to one of such convicts over their fellow workers the same as if there existed a voluntary contract of employment, but when the operator so constitutes a convict its agent and servant for certain purposes, it can not receive the benefit

of such service and at the same time exempt itself from liability for the negligence of such constituted agent.

Sloss-Sheffield Steel & Iron Co. *v.* Weir, 60 Southern, 851 (Alabama), January, 1913.

INJURY TO COAL-CAR DRIVER.

A driver of a car in a coal mine is entitled to recover for injuries caused by falling coal where the mine owner and operator failed to make proper inspection and failed to warn the driver of the danger, or properly support the roof to prevent falling of loose material, where it was no part of the driver's duty to inspect the roof of the mine or report any dangerous conditions in the roof, and where the defect and danger was not so open, patent, and of such a nature as to be discoverable by the driver passing along the entry in the discharge of his duties.

Prairie Creek Coal Mining Co. *v.* Kittrell, 153 Southwestern, 59 (Arkansas), December, 1912.

B. RIGHTS AND DUTIES OF MINER.

MINER MAKING WORKING PLACE SAFE.

The rule that it is the duty of a master to furnish safe working places for his servants applies to mine owners and operators, but the rule does not apply where a miner is employed in a mine and is making his own place to work in, and where such place is the result of the particular work for which the miner was employed, or where the place is inherently dangerous and necessarily changes as the work progresses.

Creede United Mines Co. *v.* Hawman, 127 Pacific, 924 (Colorado), November, 1912.

MINER'S WORKING PLACE.

A mine operator is not required to use ordinary care to furnish a miner a safe working place where such miner is mining coal in a room or entry in a mine and which by reason of the work of mining is constantly shifted and changed as the direct result of the mining, and especially so where a statute makes it the duty of the miner to properly prop the roof of his working place.

Proctor Coal Co. *v.* Beavers, 152 Southwestern, 965 (Kentucky), January, 1913.

MINER MAY ASSUME HIS WORKING PLACE IS SAFE.

A miner whose duty it is to drill blast holes that are charged with dynamite, for the purpose of breaking down coal in a mine, has the right to assume that the mine boss, in accordance with his duties, has examined the holes after the blast to see that all shots have been properly fired, and may return to his place of work after such mine boss has made his inspection under such assumption, and he may

recover for an injury occasioned by the failure of the mine boss to discharge this duty.

Cook *v.* Cranberry Furnace Co., 76 Southeastern, 473 (North Carolina), November, 1912.

DUTY TO KEEP ENTRY SAFE—USE OF PROPS.

The statute of Tennessee imposes upon the miner the duty to keep the roof of his working place properly propped and timbered to prevent falling rock or slate, and a failure of a miner to properly prop the roof of his working place will defeat a recovery for any consequential injury, unless it is shown that the mine operator failed to furnish the proper timbers.

Proctor Coal Co. *v.* Beavers, 152 Southwestern, 965 (Kentucky), January, 1913.

MINER'S RELIANCE ON FOREMAN'S JUDGMENT.

Where a miner was not charged with the duty of inspecting or keeping in reasonably safe condition the room or entry in which he works and had the right to rely upon the mine foreman's inspecting and keeping the place reasonably safe, the miner will not be justified in relying upon the assurance of the mine foreman that the room or entry was safe if the danger was so obvious that he knew or must have known it; but if the mine operator or the mine foreman gave the miner to understand that he did not consider the risk one which a prudent man would refuse to undertake, then the miner, notwithstanding his knowledge of the danger, may rely upon the master or foreman's judgment, and may recover in case he is injured.

Sunrise Coal Co. *v.* McDaniel, 150 Southwestern, 3 (Kentucky), October, 1912.

C. CONTRACTS RELATING TO OPERATIONS.

POWER OF MINING CORPORATION TO CONTRACT.

A zinc-mining company owning and operating mines entered into a contract by which it agreed to sell and deliver zinc concentrates at the rate of 100 tons weekly for three years, commencing at a future date. On the subsequent operation of the mines the corporation, discovering that it could not produce sufficient ore and concentrates to fulfill the contract, was authorized under its charter to purchase sufficient ore and concentrates with which to fulfill its contracts.

Young *v.* United States Zinc Cos., 198 Fed., p. 593, September, 1912.

POWER OF GENERAL MANAGER OF OIL COMPANY TO MAKE CONTRACTS.

The general manager of an oil company has implied authority to bind the company by a contract for the employment of a superintendent of the company's refineries for a period of one year, and to

agree on the compensation that should be paid him, where such general manager also has the power to discharge employees within a year for incompetency, disloyalty to the company, or insubordination in refusing to obey the orders of the general manager.

Manross *v.* Uncle Sam Oil Co., 128 Pacific, 385 (Kansas), December, 1912.

POWER OF SUPERINTENDENT OF MINING COMPANY.

An officer and superintendent of a mining company, placed in charge of the company's plant and mine by the board of directors without having his powers and duties defined, has the right as such officer or agent to enter into such contracts only as are usual and necessary to carry on the company's business according to general custom and usage of the coal-mining business in the particular territory, and whether or not a contract made by such officer or agent was a special and unusual one, or whether it was usual and necessary to carry on the business, was a question of fact to be determined by a jury.

Raven Red Ash Coal Co. *v.* Herron, 75 Southeastern, 752 (Virginia), September, 1912.

LIABILITY FOR INJURY TO INDEPENDENT CONTRACTOR.

A contractor employed by a coal company operating a mine to drive a certain entry and air course, and for which he was paid so much per yard in the heading and employed his own help and was only under the direction of the engineers of the coal company, can not recover damages of the coal company for an injury received by the fall of rock from the roof of the entry driven by him, and there is no violation of the Alabama statute where the coal company kept a sufficient supply of props and timbers to prop the roof of the mine.

Bowen *v.* Pennsylvania Coal Co., 60 Southern, 835 (Alabama), January, 1913.

OIL AND GAS LEASES—RIGHTS AND LIABILITIES OF PARTIES.

Where one of three parties engaged in the joint enterprise of procuring, improving, and handling oil and gas leases and properties caused wells to be drilled on his own land that were turned into the common holdings, the expense of such drilling must be borne by the parties as a common expense, including the expense of dry and unproductive wells.

Smith *v.* Harris, 128 Pacific, 378 (Kansas), December, 1912.

D. METHODS OF OPERATING—RIGHTS AND LIABILITIES.

STOPE—MEANING.

A stope is a working above or below a level where the mass of the ore body is broken. It is also defined to be any excavation for extraction of ore, as distinguished from a shaft, drift, airway, etc.

So a stope is said to be the very antithesis of a shaft, tunnel, drift, winze, or other similar excavation in a mine which has been finished and completed.

Creede United Mines Co. *v.* Hawman, 127 Pacific. 924 (Colorado), November, 1912.

DUTY TO VENTILATE.

The statute of Washington requires that every coal mine shall have and get sufficient ventilation, the amount of air in circulation to be not less than 100 cubic feet per minute for each man or animal in the mine, and the air must be made to circulate.

Dollar *v.* Northwestern Improvement Co., 129 Pacific, 578 (Washington), January, 1913.

It is a sufficient compliance with the statute of Illinois where a mine inspector examined a room in the mine on a morning before the miners began work and indicated the unsafe condition of the roof by chalk marks thereon, pointing out to the miner that a prop should be placed under the unsafe roof, and accordingly the miner placed a suitable prop thereunder, but this was knocked down by a blast during the day, and on the following morning before the miner commenced work the mine inspector again entered the mine, and perceiving the chalk marks on the roof were still plainly visible made chalk marks on the face of the coal, indicating the date of the inspection. Under such circumstances the mine operator was not liable to the miner injured by a fall of rock from the roof while removing the gob preparatory to setting a prop.

Kilduff *v.* Consolidated Coal Co., 99 Northeastern, 783 (Illinois), October, 1912.

DUTY OF MINE EXAMINER.

Under the statute of Illinois a mine examiner is required at all mines, and it is his duty to visit the mine before the miners are permitted to enter it and see that the air current is traveling in its proper course and in proper quantity, and for this purpose he is required to measure with a suitable instrument the amount of air passing in the last crosscut or break-through of each pair of entries or in the last room of each division in a long wall mine and at all other points where he deems it necessary, and the result of these observations must be noted in the daily book kept for that purpose, and a recovery by a miner injured by an explosion of gas because of the failure of the mine operator to comply with this statute will not be defeated because the miner passed under a canvas curtain hung at the door of the entry, where there was no sign of warning and nothing to indicate the presence of danger.

Colesar *v.* Star Coal Co., 99 Northeastern, 709 (Illinois), October, 1912.

FILLED STOPE—METHOD OF FILLING.

A "filled stope" is one where the waste rock taken out of the vein is left on the floor of the stope in order to raise the floor as the work proceeds. If there is insufficient material from the vein matter, it is the custom to blast into the side walls in order to obtain sufficient filling matter, so that the floor may be kept conveniently close to the roof to facilitate the drilling into the roof of the stope, and under such circumstances timbers, stull-timbers, or lagging are usually unnecessary, except lagging may be used to construct a crib for such convenient working.

Creede United Mines Co. *v.* Hawman, 127 Pacific, 924 (Colorado), November, 1912.

COAL WRONGFULLY MINED—MEASURE OF DAMAGES.

Where coal is wrongfully mined from another's land, but in good faith and as the result of an honest mistake, the measure of damages is the value of the coal taken as it lay in the mine, or the usual reasonable royalty paid for the right of mining, and the royalty in such case is the price paid for coal as it lies in the earth, and the fair market price is the usual standard for measuring the value; and the owner can not ask more where coal has been taken innocently under an honest mistake as to the location of the boundary line.

Burke Hollow Coal Co. *v.* Lawson, 151 Southwestern, 657 (Kentucky), December, 1912.

E. STATUTORY REGULATIONS—EFFECT AND NONOBSERVANCE.

VALIDITY OF LAWS REGULATING MINES.

A law which protects the lives, health, safety, and comfort of the miners is within the police power of a State.

Booth *v.* State, 100 Northeastern, 563 (Indiana), January, 1913.

PROSECUTIONS FOR VIOLATIONS OF MINING LAWS.

The fact that the mining act of 1905 of the State of Indiana places upon the inspector of mines the duty to see that its provisions are enforced does not necessarily limit the right to institute prosecutions for the violations of the act to the inspector or his deputies.

Malone *v.* State, 100 Northeastern, 567 (Indiana), January, 1913.

VIOLATION AS EVIDENCE OF NEGLIGENCE.

The mere violation of a statute is not even prima facie evidence of actionable negligence, except when a court can say as a matter of law that the consequences against which the statute was intended to provide have actually ensued from its violation.

Dickinson *v.* Stuart Colliery Co., 76 Southeastern, 654 (West Virginia), November, 1912.

INADVERTENT FAILURE TO COMPLY.

An action may be maintained for an injury to a miner or for the death of a miner caused by the failure of the mine operator to comply with a coal-mining statute, whether the operator's failure to comply with the statute was a mere inadvertence or a willful violation.

Peabody Alwert Coal Co. *v.* Yandell, 100 Northeastern, 758 (Indiana), February, 1913.

OPINION OF WITNESS AS TO COMPLIANCE.

In an action by a miner against a mine operator for damages for injuries caused by falling down a stairway in a shaft, due to the lack of proper banisters to the stairway, a witness can not state that in his opinion the mine was in good condition and was operated in compliance with the State mining laws, as this would involve a construction of the mining laws on the part of the witness himself; nor can he state that the entries and driveways were in a safe condition, but must state their manner of construction and leave it for the jury to determine whether the mine was properly and safely constructed.

Consumers Lignite Co. *v.* Hubner, 154 Southwestern, 249 (Texas), March, 1913.

DUTIES OF MINE BOSS.

The statutes of Indiana require a mine boss to see that every working place is properly secured by timbering and that a sufficient amount of timbers are always on hand at the miner's working place, and when an unsafe place is reported to such mine boss, then no one shall enter the place except for the purpose of making it safe; and the statutes do not contemplate that the ordinary mine work should continue in such dangerous place, but on the contrary that it should cease until the place is made safe.

Peabody Alwert Coal Co. *v.* Yandell, 100 Northeastern, 758 (Indiana), February, 1913.

FAILURE TO EMPLOY MINE BOSS.

It is a violation of the statute of West Virginia for a coal mining company to fail or refuse to employ a mine boss, and it is also a violation of the statute for a coal mining company operating its own mine to remove a mine boss and undertake itself to superintend the care of the roof of its mine and the maintenance of its safety, and it is liable to a miner injured by the fall of slate or rock from the roof of the mine permitted by the company to become and remain unsafe.

Peterson *v.* Paint Creek Collieries Co., 76 Southeastern, 664 (West Virginia), November, 1912.

DAILY INSPECTION OF MINE.

The statute of Missouri requires a daily inspection of mines generating explosive gas in which men are employed; but this statutory rule does not apply to mines not generating explosive gas, and as to

such mines the duty of the mine owner and operator is defined by the general rule requiring a master to exercise reasonable care to provide his servant a reasonably safe place in which to work.

Goode v. Central Coal & Coke Co., 151 Southwestern, 508 (Missouri), November, 1912.

FAILURE TO VENTILATE MINE.

In an action by a miner for injuries on the ground of negligence of the operator in failing to properly ventilate its mine, as required by the statute of Washington, the quantity of air necessary for the proper ventilation can not be measured by the judgment of a witness, but must be determined by the requirements of the statute.

Dollar v. Northwestern Improvement Co., 129 Pacific, 578 (Washington), January, 1913.

LIABILITY FOR PERMITTING ROADWAYS TO BECOME OBSTRUCTED.

A mine operator is liable for an injury to a miner caused by permitting dangerous obstructions to accumulate in a roadway of a mine where miners and employees are required to pass or work in performing their duties and where the permitting of such obstructions is in violation of a statute.

Eaton v. Marion County Coal Co., 101 Northeastern, 58 (Illinois), February, 1913.

REFUGE HOLES REQUIRED.

The statute of West Virginia makes it the duty of the mine owner to see that refuge holes are properly made along motor roads in coal mines.

May v. Davis Coal Co., 76 Southeastern, 342 (West Virginia), October, 1912.

DUTY TO FURNISH PROPS.

A mine operator does not comply with the statute of Kentucky imposing on him the duty of furnishing suitable props where he merely furnished such props to a youthful and inexperienced miner who was not sufficiently skilled in mining to know how to set such props, and where the operator failed either to instruct such miner or furnish an experienced miner to instruct or help him in setting the props.

Bartley v. Elkhorn Coal & Coke Co., 152 Southwestern, 965 (Kentucky), January, 1913.

FAILURE TO FURNISH TIMBERS.

A mine operator can not be held liable for failure to furnish a safe working place for a miner because of his failure to comply with the requisitions of a mine boss, as required by the statute of West Virginia, on a mere notice by such mine boss of his inability to remedy a defect and remove a "pot" or "kettle bottom" in the roof of a

haulageway, since both operator and mine boss continue under legal duties after such notice, and such a notice does not absolve the mine boss from further duty, and it was still incumbent on him, under the statute, to do the work or have it done if the mine operator furnished the materials, machinery, and labor; and to render the operator liable it must be shown that he failed, pursuant to such requisition, to furnish the necessary materials, machinery, and labor.

Peterson *v.* Paint Creek Collieries Co., 76 Southeastern, 664 (West Virginia), November, 1912.

The statute of Indiana imposes on mine operators the duty of supplying the working places of a miner with the timbers needed from time to time as the work progresses in order to make his working place secure, and the law requires the mine boss to visit the mine every alternate day, and the operator of a mine can not escape liability for injury to a miner caused by falling coal because the miner did not register on the blackboard a request for props and timbers, as provided by the statute, where the mine boss had knowledge that the necessary timbers were not supplied in the miner's working place.

Peabody Alwert Coal Co. *v.* Yandell, 100 Northeastern, 758 (Indiana), February, 1913.

MINER'S NOTICE FOR DOUBLE TIMBERING.

The statute of Iowa makes it a misdemeanor for a miner to neglect or refuse to securely prop or support the roof and entries under his control; but where the coal operators and the mine workers entered into an agreement to the effect that where a miner has properly timbered his room or entry and the roof from any cause becomes so heavy as to require double timbering, then the mine operator shall, when notified by the miner, do the necessary work to protect the roadway, and a miner is not subject to a charge of a violation of the statute and is not guilty of contributory negligence where he had securely propped the roof and had notified the mine operator that double timbering was required.

Braddich *v.* Phillips Coal Co., 138 Northwestern, 406 (Iowa), November, 1912.

LIABILITY OF OPERATOR FOR INJURIES FROM FALLING COAL.

An allegation in a complaint in an action by a miner for injuries caused by falling coal, to the effect that the mine boss knew that the roof was insecure and defective and in a condition to fall at any time, and that such mine boss had such knowledge for a period of from two to six days preceding the injury and in time to have taken down the loose material that caused the injury, is sufficient as an averment of the actual dangerous condition of the roof.

Domestic Block Coal Co. *v.* De Armey, 100 Northeastern, 675 (Indiana), January, 1913.

DUTIES OF OPERATOR TO REMOVE LOOSE MATERIAL.

The Indiana statute of 1897 providing that all loose coal, slate, and rock overhead, where miners have to travel to and from their work, must be carefully secured, was amended by the act of 1905 by the addition of a provision that such loose coal, slate, or rock should be taken down or carefully secured, and the purpose of the amendment was to compel the removal of loose material in places where it was impracticable to secure the same by means of props and timbers, and the provision includes a way or track in a room in which miners are working as well as travel ways through the mine.

Domestic Block Coal Co. v. De Armey, 100 Northeastern, 675 (Indiana), January, 1913.

WORKING MORE THAN EIGHT HOURS A DAY.

The statute of Montana prohibits a mine owner from employing miners to work more than eight hours a day in a mine and also prohibits miners from working more than eight hours a day, and a violation of the statute is legal negligence, and if a violation of the law on the part of a mine operator is negligence and the proximate cause of an injury resulting to a miner, so the disobedience of the statute by the miner himself must be regarded as a contributing proximate cause, and the disobedience is concurrent and the injury is the result of the concurrent causes operating to the same end, and the miner can not recover, for the reason that in alleging the injury he must necessarily show his own contributing fault.

Melville v. Butte-Balalava Copper Co., 130 Pacific, 441 (Montana), February, 1913.

PERSON SERVING AS HOISTING ENGINEER WITHOUT CERTIFICATE.

An indictment against a person under the statute of Indiana for serving as a hoisting engineer in a coal mine without a certificate of competency is not required to negative the statutory provision that fewer than 10 men were employed in the mine, as that is a matter of defense.

Malone v. State, 100 Northeastern, 567 (Indiana), January, 1913.

EMPLOYMENT OF BOY IN MINE.

The statute of Alabama providing that no woman, and no boy under the age of 14 years, shall be employed in a mine was intended to protect women and children of tender age from incurring the hazard and danger incident to the operation of mines by absolutely preventing their employment; and the statute requires the mine owner or operator to see and know that those employed are not within the prohibited class, and the statute must be liberally construed, so as to effectuate the humane intent of the legislature.

De Soto Coal Mining & Development Co. v. Hill, 60 Southern, 583 (Alabama), December, 1912.

EMPLOYMENT OF BOY—LIABILITY FOR INJURY.

The employment of a boy under the age of 14 years in a coal mine is a violation of the statute and constitutes actionable negligence whenever such violation is the natural and proximate cause of the injury, though the boy himself may have negligently contributed to his own injury, if such contributory negligence was of that character for which the statute was intended with respect to boys of that age, for the reason that boys employed in violation of a statute can not be regarded as having assumed the risks of any injuries resulting from the ordinary operations of a coal mine.

Dickinson *v.* Stuart Colliery Co., 76 Southeastern, 654 (West Virginia), November, 1912.

INJURY TO BOY OF PROHIBITED AGE.

A mining company employing a boy under 14 years of age to work in its mine in violation of the statute of Alabama is liable to him for injuries resulting by reason of such employment, and in such case it is not a question as to whether or not the mining company believed the boy was over the prohibited age, nor is it necessary that injury must result as a proximate cause of some act or omission of the boy in the discharge of his duty; but the right of action arises where the injury resulted from the employment and was incident to any of the risks or dangers in or about the business, and this liability is not excused by reason of false representations by the boy or anyone else as to his age, and such representations do not estop him from a recovery for the injuries sustained.

De Soto Mining & Development Co. *v.* Hill, 60 Southern, 583 (Alabama), December, 1912.

EMPLOYMENT OF BOY WITH FATHER'S CONSENT.

Where a father was a party to a contract of employment of his son under 12 years of age in a coal mine, in violation of the statute of West Virginia, and the boy was killed by an explosion in the mine and the father was the sole beneficiary of any recovery, the contributory negligence of the father in respect to such employment will prevent his recovery against the mine operator for damages unless such owner or operator was himself guilty of some act of negligence that was the proximate cause of the explosion and the consequential death of the boy.

Dickinson *v.* Stuart Colliery Co., 76 Southeastern, 654 (West Virginia), November, 1912.

VALIDITY OF STATUTES REQUIRING WASHHOUSES.

The statute of Indiana requiring owners or operators of coal mines and other employers of labor to erect and maintain washhouses is valid, as being within the police power of the State, and includes

superintendents within its scope; and a superintendent of a mine who fails, on the written request of the employees of the mine, to provide and maintain a washhouse for the use of the miners is subject to the penalty imposed by the statute.

Booth *v.* State, 100 Northeastern, 563 (Indiana), January, 1913.

MINE OPERATORS REQUIRED BY STATUTE TO MAINTAIN WASHHOUSES.

A statute requiring an owner or operator of a coal mine where more than 20 persons are employed to construct and maintain washhouses is not unconstitutional in that it denies to coal companies the equal protection of the law and discriminates between coal mining and other classes of business, or deprives the mine owner of his property without due process of law, as such a statute falls within the police power of the State.

Booth *v.* State, 100 Northeastern, 563 (Indiana), January, 1913.

POLLUTION OF WATER.

Under the statute of West Virginia making it an offense to put into a stream of water any matter deleterious to the propagation of fish, an operator of a coal mine can not assert a right contrary to the statute to drain into a stream from his mine sulphur or mine water deleterious to the propagation of fish, though such stream is the natural receptacle for such drainage and it is impracticable to drain the mine otherwise.

State *v.* Southern Coal & Transportation Co., 76 Southern, 970 (West Virginia), December, 1912.

F. NEGLIGENCE OF OPERATOR—LIABILITY.

EXPLOSION AS EVIDENCE OF NEGLIGENCE.

The fact of an explosion in a coal mine is not even prima facie evidence of negligence on the part of the mine foreman.

Dickinson *v.* Stuart Colliery Co., 76 Southeastern, 654 (West Virginia), November, 1912.

EXPLOSIONS—JUDICIAL NOTICE.

Courts take judicial notice that explosions occur in the best-equipped, best-regulated, and perfectly ventilated coal mines, and a mine owner or operator is not liable for resulting injuries when an explosion is the result of negligence of a fellow servant, and not of the owner or operator to furnish a reasonably safe place in the first instance or to employ a competent mine boss or to perform some other statutory or common-law duty imposed.

Dickinson *v.* Stuart Colliery Co., 76 Southeastern, 654 (West Virginia), November, 1912.

PLEADING NEGLIGENCE AND PLACE OF INJURY.

A complaint in an action by a miner for injuries caused by the negligence of a coal-mine operator can not be regarded as defective and can not be held insufficient for failing to specify the exact location in the mine where the accident happened, where the particular place is not a material feature of the negligence complained of.

Birmingham Coal & Iron Co. *v.* Brice, 60 Southern, 952 (Alabama), January, 1913.

PROXIMATE CAUSE OF INJURY.

A court can not say as a matter of law that permitting obstructions in violation of a statute, in an entryway of a mine where a miner was required to work, was not the proximate cause of an injury, where such obstructions were permitted at a steep incline, rendering it necessary to sprag cars, and where the miner was, while endeavoring to sprag the wheels of a car, by reason of the dangerous conditions caused by such obstructions, caught between the car and the rib of the mine.

Eaton *v.* Marion County Coal Co., 101 Northeastern, 58 (Illinois), February, 1913.

FAILURE TO EMPLOY MINE BOSS.

A coal-mining company that ousts a mine boss after his employment, in compliance with the statute of West Virginia, and itself undertakes the care of the roof of its mine, and then permits the roof to become and remain unsafe and dangerous, is liable to a miner injured by a fall of slate or stone from the roof, as the negligence in such case is not negligence of the mine boss, but that of the mine owner and operator.

Peterson *v.* Paint Creek Collieries Co., 76 Southeastern, 664 (West Virginia), November, 1912.

MINERS FOLLOWING DIRECTIONS OF MINE FOREMAN.

A coal company operating a coal mine is liable for an injury of a miner caused by falling coal or slate where the miner at the time of the injury was following the direction of the mine foreman, and was doing what he told him to do and in the particular way he told him to do it, and where the miner himself did not create the danger in the progress of his work, and where the danger existed at the time the mine foreman directed the work to be done and was incidental to the work so directed.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

FAILURE TO INSTRUCT INEXPERIENCED MINER.

A mine operator can not be charged with negligence for failure to instruct an inexperienced miner that loose coal might be discovered by tapping, where such failure to instruct and where the want of tapping the coal was not the proximate cause of an injury.

Carney Coal Co. *v.* Benedict, 129 Pacific, 1024 (Wyoming), February, 1913.

NOTICE TO MINE FOREMAN.

Knowledge of the fire boss and the mine foreman that a "squeeze" or settling of the roof had set in, was notice to the company operating the mine.

Gennaux *v.* Northwestern Improvement Co., 130 Pacific, 495 (Washington), February, 1913.

DUTY TO INSPECT MINE.

A mine operator is bound to inspect at reasonable intervals the roof of a mine where miners are at work, and to remove any insecure rock or other material that threatens the safety of the miners whose service compels them to use the particular part of the mine; and a failure of a mine operator to inspect the roof at reasonable intervals and where reasonable inspection would have disclosed any existing danger is sufficient to sustain a charge of negligence, and to show that such failure to inspect was the proximate cause of an injury to a miner.

Goode *v.* Central Coal & Coke Co., 151 Southwestern, 508 (Missouri), November, 1912.

INSPECTION BEFORE MINERS BEGIN WORK.

The statute of Illinois requires a mine owner or operator to visit the mine before miners are permitted to enter and see that the air current is traveling in its proper course and in proper quantity, to determine the quantity of air in different portions of the mine and the amount of air passing in the last crosscut or break-through in each pair of entries as well as in the last room of each division, and to note the condition of the mine in a book kept for that purpose before miners are permitted to enter the shaft; and the failure to make such examination and to inform the miners of the presence of gas is such negligence as will render the mine owner and operator liable for an injury to a miner occasioned by an explosion of gas in the entry or room in which he was working.

Colesar *v.* Star Coal Co., 99 Northwestern, 709 (Illinois), October, 1912.

FAILURE TO INSPECT MINE.

It is actionable negligence for a mine owner or operator through his mine boss to fail to inspect properly the places where blasting is done and to examine after each blast the holes charged with dynamite

to see if any have failed to go off, and in case of any such failure, to give proper notice to the miners drilling the blast holes, and any such mine owner or operator is liable to an injured miner for negligence in this respect.

Cook *v.* Cranberry Furnace Co., 76 Southeastern, 473 (North Carolina), November, 1912.

A coal-mine operator was held liable for damages to a driver caused by a fall of slate where the operator negligently failed to inspect a place where coal had been loosened and to remove the loose coal, or to place timbers thereunder or notify the driver of the dangerous condition, and where it was not the duty of the driver to inspect the mine for loose coal, and such negligent failure on the part of the mine operator to properly inspect the mine was held to be the proximate cause of the injury.

Prairie Creek Coal Mining Co. *v.* Kittrell, 153 Southwestern, 59 (Arkansas), December, 1912.

FAILURE TO FURNISH SAFETY LAMPS.

The Legislature of Washington in recognition of the hazards of working in coal mines has made careful provisions for their inspection and imposed imperative duties upon those who own and operate them and requires among other things that under certain circumstances safety lamps shall be furnished the miners, and a failure to observe this provision is negligence per se; but a failure to furnish such safety lamps is not negligence in the absence of evidence showing that the development work being prosecuted was approaching a place where there was likely to be an accumulation of excessive gases or that there was imminent danger from excessive gases.

Dollar *v.* Northwestern Improvement Co., 129 Pacific, 578, p. 580 (Washington), January, 1913.

FAILURE TO PROP ROOF.

The question of the negligence of a mine owner to properly support the roof of a crosscut in which the miners were not required to work, but in which a miner was injured by a falling of roof while eating a meal, is one of fact to be determined by the jury under all the circumstances of the case.

Kennis *v.* Ogden, 138 Northwestern, 467 (Iowa), November, 1912.

FAILURE TO FURNISH PROPS.

The statute of Kentucky imposes on the owner or operator of the mine the duty of furnishing suitable props for securing the roof and for making the working place of the miner safe. A failure to

supply such props is such a breach of this statutory duty as will render the operator liable for any resulting injury to a miner.

Bartley *v.* Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1913.

FAILURE TO SUPPORT ROOF AFTER NOTICE.

A mining company is liable for negligence in permitting the roof of an entry in its mine to become unsafe, and for failure to properly support the roof after notice of its dangerous condition, where by reason of such negligence a miner was injured by the fall of the roof after the mine foreman had promised to remedy its dangerous condition, and where the miner was carrying out the immediate directions and orders of the mine foreman.

Broadway Coal Mining Co. *v.* Robinson, 150 Southwestern, 1000 (Kentucky), November, 1912.

PROOF OF NEGLIGENCE—INJURY FROM FALLING ROCK.

In an action for the death of a miner caused by a fall of rock from the roof of a mine at a place that was not his working place, a court is not authorized to direct a jury to return a verdict for the plaintiff without reference to the issues of whether or not the dangerous character of the rock would have been discoverable to an ordinarily careful and prudent mine operator in the exercise of due care toward his miner, and whether or not the mine operator did employ due care in the discharge of his duty in this respect. Under such circumstances the fact that rock fell of itself was not sufficient proof of negligence, and the burden of proof in such case was on the plaintiff to satisfy the jury that the rock not only was in a dangerous condition but that such condition would have been discovered by the mine operator had reasonable care been exercised in time to have prevented the injury.

Goode *v.* Central Coal & Coke Co., 151 Southwestern, 508 (Missouri), November, 1912.

FAILURE TO SUPPLY MACHINERY.

The failure of a mining company operating a coal mine to furnish its mine boss with machinery and materials necessary to remove any dangerous slate or stone in the roof of the mine and otherwise make it secure, upon notice by the mine boss of his inability to make the mine roof safe on account of lack of materials or facilities, is such negligence that renders it liable for an injury to an employee caused by a fall of such loose slate or stone.

Peterson *v.* Paint Creek Collieries Co., 76 Southeastern, 664 (West Virginia), November, 1912.

USING DEFECTIVE MACHINERY.

A coal-mine operator is guilty of actionable negligence in using a worn and dangerous feed chain used in drawing forward a coal-mining machine, with knowledge of its condition and by reason of which a miner was injured.

Bradley *v.* Northern Central Coal Co., 151 Southwestern, 180 (Missouri), November, 1912.

FAILURE TO REMOVE LOOSE COAL OR ROCK.

Where it becomes impracticable to secure loose coal, slate, or rock overhead in a working place in a mine by means of timbers or props, then it is the duty of the mine operator to have such loose coal, slate, or rock removed before the miners are permitted to resume work, and the failure to do so will constitute negligence.

Peabody Alwert Coal Co. *v.* Yandell, 100 Northeastern, 758 (Indiana), February, 1913.
Domestic Block Coal Co. *v.* De Armey, 100 Northeastern, 675 (Indiana), January, 1913.

FAILURE TO MAKE REFUGE HOLES.

A mine owner or operator is not chargeable with actionable negligence for an injury resulting to a miner because of his failure to make refuge holes along motor roads in a coal mine, as the statute imposes this duty on the mine foreman and not on the mine operator.

May *v.* Davis Coal Co., 76 Southeastern, 342 (West Virginia), October, 1912.

INJURY TO MINER BY ELECTRIC WIRE.

A court can not say as a matter of law that a miner is not entitled to recover for an injury caused by contact with an unprotected trolley wire where the miner was engaged in carrying a rubber hose bound with wire, and where, by reason of the wet and slippery condition of the floor of a level, he slipped and the wire-bound hose accidentally came in contact with the live wire and the miner thereby received the injury for which he sues.

Williams *v.* Bunker Hill & Sullivan, etc., Mining Co., 200 Fed., 211, October, 1912.

OPERATING MINE CAR AT DANGEROUS SPEED.

It is negligence for a mine operator to run a car carrying a miner through a slope with a low and dangerous roof at such a rate of speed as to extinguish the light on the miner's cap and leave him in the dark, thereby unable to distinguish such low and dangerous place in the roof, by reason of which he was injured; and the movement of the car with such undue speed may be regarded as the efficient proximate cause operating in unbroken sequence to bring about the injury.

Republic Iron & Steel Co. *v.* Fuller, 60 Southern, 475 (Alabama), December, 1912.

INJURY TO ROPE DRIVER—SCOPE OF EMPLOYMENT.

It was the duty of a rope driver in a coal mine to look after the rope and the equipment of the train of cars drawing coal from the mine and to remedy any trouble that might come up in connection with the operation of the train, and such rope driver was acting within the line of his employment and was doing what his duty required him to do when he attempted to stop a train of coal cars for the purpose of releasing the rope which had caught under the end of some of the exposed ties supporting the track; and the coal company was held liable for an injury occurring to the rope driver while thus performing his duty, because of its negligence in permitting the ties to be so laid and exposed as to catch and hold the rope.

Stony Fork Coal Co. *v.* Lingar, 153 Southwestern, 6 (Kentucky), February, 1913.

NEGLIGENCE OF FELLOW SERVANT.

A rope driver working on cars used to haul coal out of a mine whose duty it is to superintend the movement of cars in and out of a mine and to fasten and unfasten the cable by which the cars are operated and the engineer employed to operate the stationary engine that furnished power to move the cars on signals from such rope driver are fellow servants, and such rope driver can not recover from the mine operator for injuries caused by the negligence of the engineer.

Bell-Knox Coal Co. *v.* Gregory, 153 Southwestern, 465 (Kentucky), February, 1913.

WRONG SIGNAL TO MINER.

A coal operator is liable for damages for the death of a miner caused by a rock from blasting where the foreman negligently called to the employees to return to work by hallooing "all over," meaning as was the custom that the shots had all been fired, and the deceased miner in response to this customary signal started from his safe hiding place to return to work, when a rock from the second blast struck and killed him.

Chenoa-Hignite Coal Co. *v.* Philpot, 153 Southwestern, 457 (Kentucky), February, 1913.

PROOF OF SUBSEQUENT REPAIRS NOT ADMISSIBLE.

In an action against a coal-mining company for the death of a miner caused by an alleged insecure platform under a tipple used in connection with a coal screen, proof of repairs made upon the platform subsequent to the time of the accident is not admissible for the purpose of proving negligence and to show that the platform was insecure at the time of the accident.

Interstate Coal Co. *v.* Shelton, 153 Southwestern, 1 (Kentucky), February, 1913.
Bell-Knox Coal Co. *v.* Gregory, 153 Southwestern, 465 (Kentucky), February, 1913.

PROOF OF SUBSEQUENT REPAIRS—EXCEPTION.

While ordinarily proof of subsequent repairs is not admissible for the purpose of showing negligence yet where in an action by a rope driver working on cars hauling coal out of a mine he introduced evidence to show the defective and dangerous condition of the track, and where the mine operator introduced evidence to show that the person who succeeded the injured rope driver in his position continued the work without accident and without any change being made in the tracks at the place where the injury occurred, the injured rope driver then has the right to show as a matter of fact that the track was repaired subsequent to the time of his injury and at the particular place where the accident happened.

Bell-Knox Coal Co. *v.* Gregory, 153 Southwestern, 465 (Kentucky), February, 1913.

LIABILITY FOR BURIAL EXPENSES.

Under the statute of Oklahoma the duty of burying the dead bodies of miners whose lives were lost in a mine disaster is cast upon the mine owner where it does not appear that they had relatives or friends present to perform this duty, and where no person in fact claimed the bodies.

Kali Inla Coal Co. *v.* Craig, 128 Pacific, 117 (Oklahoma), November, 1912.

G. CONTRIBUTORY NEGLIGENCE OF MINER.

VOLUNTARY EXPOSURE TO DANGER.

A miner can not maintain an action against a mine operator for damages for an injury resulting from a known peril to which he voluntarily continued to expose himself.

Republic Iron & Steel Co. *v.* Fuller, 60 Southern, 475 (Alabama), December, 1912.

KNOWLEDGE OF DANGER.

A driver of coal cars in a mine is not entitled to recover for injuries received from falling coal where the dangerous condition of the coal was open, patent, or of such a nature as to be discoverable by any driver passing along the entry in the discharge of his duty, and if by the use of his eyes, or by ordinary care, he could have seen the danger, and if he knew and apprehended the dangers and failed to exercise ordinary care in avoiding the same.

Prairie Creek Coal Mining Co. *v.* Kittrell, 153 Southwestern, 59 (Arkansas), December, 1912.

KNOWLEDGE OF DEFECTIVE APPLIANCES.

A miner operating a coal-mining machine with knowledge that the feed chain used in drawing the machine forward was defective is not to be charged with contributory negligence in case of an injury, as

under such circumstances he had the right to continue in the service, provided a reasonably careful person in his situation would have concluded that he could work with the chain without subjecting himself to danger of immediate injury.

Bradley v. Northern Central Coal Co., 151 Southwestern, 180 (Missouri), November, 1912.

AVOIDING DANGER IN EMERGENCY.

A miner exposed to a danger that by reason of another's negligence is suddenly enhanced by an unfavorable change of the conditions under which it is to be met, due to such negligence, is not necessarily chargeable with contributory negligence because he did not exercise the best judgment in the measures taken for his own safety, as the law does not under such circumstances hold him infallible in his endeavor to escape harm, and his conduct is to be weighed in the light of his surroundings at the time and he is to be held to no more than the exercise of such reasonable care and diligence as would be expected of a prudent and reasonable man under similar circumstances.

Republic Iron & Steel Co. v. Fuller, 60 Southern, 475 (Alabama), December, 1912.

BURDEN OF PROOF.

The burden of proof is on a mine operator to show that a youthful and inexperienced miner failed to use ordinary care for his own safety, but for which he would not have been injured.

Bartley v. Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1913.

DEFENSE AND BURDEN OF PROOF.

Under the statute of North Carolina the contributory negligence of a miner is a matter of defense, and in an action for personal injury to a miner the burden is on the mine owner or operator to establish such contributory negligence, unless the testimony of the plaintiff himself shows that his contributory negligence was the proximate cause of his injury.

Cook v. Cranberry Furnace Co., 76 Southeastern, 473 (North Carolina), November, 1912.

MITIGATION OF DAMAGES.

The statute of Tennessee makes it a misdemeanor for a miner to fail to keep the roof of his working place properly propped and timbered to prevent the falling of coal, slate, or rock. Under the statute, if a miner was guilty of negligence that contributed to his injury, but if the mine operator was guilty of negligence which was the direct or proximate cause of the injury, then the miner may recover, but the damages recoverable should be reduced or mitigated by reason of the miner's contributory negligence.

Proctor Coal Co. v. Beaver, 152 Southwestern, 965 (Kentucky), January, 1913.

VIOLATION OF ORDERS.

A miner whose duty it is to drill blast holes is guilty of contributory negligence in drilling into a failed hole in violation of the orders and directions of the mine boss, and can not recover for an injury occasioned thereby, where he knew it was the duty of the mine boss to examine the blast holes after each shot to ascertain if there was any exploded charge, and where he was not to resume work until the mine boss had so reported, and in case of any unexploded charge until such charge was exploded or removed.

Cook *v.* Cranberry Furnace Co., 76 Southeastern, 473 (North Carolina), November, 1912.

DISOBEYING FOREMAN'S ORDERS.

A miner engaged in mining out pillars in a coal mine can not recover for an injury caused by a fall of slate from the roof if he violated the orders of the mine foreman or did not perform the work in the manner in which he was directed to perform, or if it was his duty to prop the roof of the mine at the place and failed to do so.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

OBEDIENCE TO FOREMAN'S ORDERS.

A coal miner with no experience in shaft mines can not be charged with such contributory negligence as would defeat a recovery for injuries where he was ordered by the foreman to go to another part of the mine and where obedience to such orders made it necessary that he should pass through the elevator shaft, and where, before passing through the shaft, he looked up the shaft but could neither see nor hear the descending elevator, but was injured by an elevator while passing through the shaft.

Fluehart Collieries Co. *v.* Elam, 151 Southwestern, 34 (Kentucky), December, 1912.

COMPLIANCE WITH CUSTOM.

A miner assisting in sinking a prospecting shaft is not to be charged with contributory negligence as a matter of law because he was riding back from the dump on the car used to haul away the excavated earth, or because he stepped on the car to attach the rope to the hoisting bucket, and thereby the car ran over the block or bumper on the track and threw him into the shaft, unless the act of riding or getting on the car was necessarily and inevitably so dangerous that a prudent person would not have done the particular act. And under such circumstances, evidence showing he complied with a custom known to and expressly or impliedly sanctioned by his employer is generally sufficient to turn the scale in this respect in his favor, and free him from the charge of contributory negligence.

Killeen *v.* Barnes-King Development Co., 127 Pacific, 89 (Montana), October, 1912.

CHOOSING DANGEROUS METHOD OF WORKING.

The rule that where a servant has two ways of doing a thing, one of which is dangerous and the other safe, and chooses the dangerous and is injured by reason thereof, he is guilty of such contributory negligence as prevents recovery, is applicable to miners; but the rule does not apply unless the miner knew, or ought to have known, that one method was dangerous and the other safe. And a circumstance which tends to negative the inference that the miner was guilty of contributory negligence is that the course of action which he was pursuing at the time he received the injury was the one customarily followed by himself and other miners under similar circumstances with the approval or tacit acquiescence of the employer.

Killeen *v.* Barnes-King Development Co., 127 Pacific, 89 (Montana), October, 1912.

INJURY TO BOY OF PROHIBITED AGE.

A mining company employing a boy to work in its coal mine, in violation of a specific statute, can not screen itself from liability for an injury sustained by such boy in its service on the ground that the injury was occasioned through such negligence, imprudence, or childish traits as gave rise to the statute, as a boy so employed and injured can not, under the law, be chargeable with negligence contributing to his injury.

De Soto Coal Mining & Development Co. *v.* Hill, 60 Southern, 583 (Alabama), December, 1912.

MINER USING EXCESSIVE BLAST.

Under the Tennessee statute making it a misdemeanor for a miner to fail to properly prop the roof of his working place, a miner is guilty of such contributory negligence as will defeat his recovery for injuries caused by falling rock where he used so large a charge of powder in shooting the coal as to knock down the props and dislodge the slate in the roof.

Proctor Coal Co. *v.* Beaver, 152 Southwestern, 965 (Kentucky), January, 1913.

FALL OF SLATE FROM ROOF.

A mine owner and operator is not guilty of actionable negligence and is not liable to a miner injured by a fall of slate from the roof of an entry where the miner himself with another was engaged in timbering the entry, and where it was their duty to discover and determine the particular part of the entry or roof that was dangerous and that required timbering.

Owens *v.* Norwood White Coal Co., 138 Northwestern, 483 (Iowa), November, 1912.

A miner receiving injuries from the falling roof of a mine is not to be charged with contributory negligence where he had had but slight

acquaintance with the mine, and did not really know whether or not the loose slate would fall; and a court can not as a matter of law say that the danger was so imminent to the miner that he should not have continued work, especially after the pit boss, with his experience and superior knowledge of the particular mine, had assured the miner that the place was safe and ordered him to go on with his work.

Braddich *v.* Phillips Coal Co., 138 Northwestern, 406 (Iowa), November, 1912.

MINER'S FAILURE TO PROP ROOF.

The statute of Tennessee makes it a misdemeanor for a miner to fail to properly prop the roof of his working place, and the failure of a miner to comply with this statute will prevent his recovery for a consequential injury when the action is brought in the courts of Kentucky if, but for his negligence, the injury would not have occurred; and the Tennessee rule that contributory negligence may be applied to the mitigation of damages does not obtain in Kentucky.

Proctor Coal Co. *v.* Beaver, 152 Southwestern, 965 (Kentucky), January, 1913.

MINER KILLED BY TROLLEY WIRE.

In an action against a company operating a coal mine for damages for the death of a miner caused by coming in contact with a live trolley wire where it appeared that the wire had become detached from a clamp holding it to the wall of the mine and the miner while attempting to replace the wire under the clamp accidentally came in touch with the wire, the question of the contributory negligence of the miner under such circumstances was one of fact to be determined by a jury.

Meola *v.* Quincy Mining Co., 140 Northwestern, 460 (Michigan), March, 1913.

LIABILITY FOR DEATH OF CONVICT WORKING IN MINE.

Where a complaint in an action against a coal mine operator for damages for wrongful death charged that the plaintiff's intestate was being worked as a convict under an unsupported roof in a mine, an answer setting up contributory negligence must go beyond a mere averment of negligence as a conclusion, and must aver facts to which the law attaches that conclusion, as the ordinary rule of contributory negligence does not apply to a convict miner.

Sloss-Sheffield Steel & Iron Co. *v.* Weir, 60 Southern, 851 (Alabama), January, 1913.

MULE DRIVER INJURED WHILE NOT IN LINE OF DUTY.

A boy employed as a mule driver can not recover of a coal mine operator, where his complaint charged that he was injured while engaged in and about the discharge of his duties as such mule driver

in the mine, where the evidence showed that his duty as a mule boy was to pull empty cars from the mouth of an entry to the head of it to be loaded by the miners, and where the evidence also showed that on the occasion of the accident he was standing by the side of a slope at one end of the entries when five loaded cars in charge of the chainers passed out of an entry and up the slope, and while plaintiff was stooping down in obeying directions of a chainer to shut the latch for him he was struck by two cars that had broken loose from the train and dashed down the mine slope, as this constituted a fatal variance between the pleading and the proof.

Maxie *v.* Sloss-Sheffield Steel & Iron Co., 61 Southern, 269 (Alabama), February, 1913.

INJURY TO ROPE DRIVER.

A court can not say as a matter of law that a rope driver superintending the movements of cars in and out of a coal mine and fastening and unfastening the cable by which the cars are operated was guilty of contributory negligence because at the time of the injury he was riding with one foot in the car and the other foot on the bumper, where there was some evidence to the effect that such a position was safer than if the rope driver rode with both feet in the car, and under such circumstances the question of contributory negligence is one of fact to be determined by a jury.

Bell-Knox Coal Co. *v.* Gregory, 153 Southwestern, 465 (Kentucky), February, 1913.

INJURY FROM BLASTING—PROPER PLACE OF CONCEALMENT.

An employee in a mine, killed from being struck by a rock from a blast, was not chargeable with contributory negligence as a matter of law in concealing himself or seeking shelter behind a tree when warned of an approaching blast, instead of taking refuge in an airway, where it was his custom and that of other men working with him at a distance of two or three hundred yards from the blasting to take refuge behind trees and where the tree behind which he stood was 167 feet from the place of the blast and was of greater diameter than his body.

Chenoa-Hignite Coal Co. *v.* Philpot, 153 Southwestern, 457 (Kentucky), February, 1913.

H. ASSUMPTION OF RISK BY MINER.

1. RISKS ASSUMED.

BURDEN OF PROOF.

The burden of proof is on a mine operator to show that a youthful and inexperienced miner took the risk of injury from falling rock or slate with knowledge of the danger.

Bartley *v.* Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1913.

KNOWLEDGE OF DANGER.

An inexperienced miner 23 years of age is chargeable with knowledge of the law of gravitation and the liability of loose coal to fall, and when he and a coemployee fired a shot to loosen coal he assumed the risk of injury from the fall of coal loosened by such shot where the danger was obvious.

Carney Coal Co. *v.* Benedict, 129 Pacific, 1024 (Wyoming), February, 1913.

MINER INJURED BY FALL OF ROOF.

An employee and miner in a coal mine assumes the risks and dangers of the employment which he knows and appreciates, and he also assumes those which an ordinarily prudent person of his capacity and intelligence would have known and appreciated in his situation.

Kennis *v.* Ogden, 138 Northwestern, 467 (Iowa), November, 1912.

A miner employed to assist another in inspecting and timbering the roof of an entry in a mine, and in making such entry and the roof safe for other miners, assumes the risk naturally and necessarily incident to such work.

Owens *v.* Norwood White Coal Co., 138 Northwestern, 483 (Iowa), November, 1912.

PROMISE OF MINE FOREMAN.

An experienced miner who reported to a mine for work pursuant to a previous promise made by the mine foreman at some distance from the mine to the effect that he would be given work in a safe place, and who was, after several days, put to work in a stope, the character of which was constantly changing, and the dangerous condition of which was well known to the miner, assumed the risk, and could not recover for an injury subsequently received while working in the stope, notwithstanding the promise of the mine foreman.

Creede United Mines Co. *v.* Hawman, 127 Pacific, 924 (Colorado), November, 1912.

NATURE AND USE OF EXPLOSIVES.

A miner is presumed to know that unusual dangers attend him while working in a stope; and he will likewise be presumed to know that high explosives used therein, such as dynamite and giant powder, not only displace rock, ore, and other mineral matter adjacent to the shot, but also have a tendency to, and do, loosen other rock and material in the vein in places more or less remote from the point where the shot is fired.

Creede United Mines Co. *v.* Hawman, 127 Pacific, 924 (Colorado), November, 1912.

DANGERS FROM ELECTRICITY.

Where tram cars in a mine were operated by electric motor receiving its power from a copper wire attached to the top or hanging walls by means of an insulated swivel held by a clamp and where if at any

time the wire was out of order for any reason the miners and the motorman were required to push the cars by hand past the defective place in the wire, and a miner who, on finding the wire defective and released from the clamp, instead of pushing the car by hand past the defective place, attempted to move the car past such defective place by means of the electric power, assumed the risk of all danger of coming in contact with the trolley wire.

Meola *v.* Quincy Mining Co., 140 Northwestern, 460 (Michigan), March, 1913.

DANGEROUS EMPLOYMENT—MINING OUT PILLARS.

The work of mining out pillars or "pulling stumps" in a coal mine is a hazardous employment, and a miner accepting such employment assumes all the risks incident to the work not due to the negligence of the company operating the mine.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

2. RISKS NOT ASSUMED.

KNOWLEDGE OF DANGER.

A miner can not, as a matter of law, be charged with the assumption of risk of danger from a falling roof where he was but slightly acquainted with the particular mine, though he knew that the roof sounded "loose and drummy," but where the pit boss assured him that he knew from his greater experience that the place was safe, and thereupon ordered him to continue work, unless the danger was so apparent that a reasonably cautious man would have seen and appreciated it.

Braddich *v.* Phillips Coal Co., 138 Northwestern, 406 (Iowa), November, 1912.

WANT OF KNOWLEDGE OF DANGER.

A miner who, by reason of his youth and inexperience, is unacquainted with the dangers incident to his employment, does not assume the risk thereof where the mine operator has failed to warn him of the danger to which he is exposed and instruct him how to do his work and avoid the danger.

Bartley *v.* Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1913.

KNOWLEDGE OF DANGER—NEGLIGENCE.

A miner familiar with all parts of a mine, and with knowledge of the danger of a low place in the roof of a slope, assumed the risk of being injured in riding on a car under such dangerous low place, but his continuance in the work with full knowledge of the existence of such danger did not have the effect of a voluntary assumption by him of the further peril involved in the improper or negligent opera-

tion of the car on which he was carried; and by consenting to be carried and assuming the danger of the low place in the roof, he did not consent to be carried at a speed too great for the safety of such a trip and such dangerous place.

Republic Iron & Steel Co. *v.* Fuller, 60 Southern, 475 (Alabama), December, 1912.

PRESUMPTION AS TO SAFETY.

A driver of coal cars hauling coal out of a mine has the right to presume that the coal mine operator has discharged his duty to him in making an entry a reasonably safe place in which to perform his duties, and as such driver he was not required to inspect the road or side of the entry, or to search for defects therein, but was only required to use ordinary care for his own safety, being such care as a reasonably prudent person would exercise under like circumstances and conditions.

Prairie Creek Coal Mining Co. *v.* Kittrell, 153 Southwestern, 59 (Arkansas), December, 1912.

PRESUMPTION AS TO SAFE CONDUCT OF BUSINESS.

When a miner enters the employment of a mine operator he is not required to use ordinary care to see that the operator's business is conducted in a reasonably safe manner and he does not assume the risks arising from the failure of the mine operator to do its duty unless he knows of its failure and the attendant risks, or where in the ordinary discharge of his own duty he must necessarily acquire such knowledge.

Consumers Lignite Co. *v.* Hubner, 154 Southwestern, 249 (Texas), March, 1913.

DANGERS NOT DISCOVERABLE.

An experienced miner while engaged for the first time in driving a coal car drawn by a mule in and out of a mine is not as a matter of law to be charged with the assumption of the risk of danger from a rock protruding from the rib to a point directly perpendicular over the rail and but slightly above the edge of the car, and where his position as driver was not favorable to the discovery of the particular danger, and where the rock protruding was not so obvious that the miner was bound to know it.

Ek *v.* Phillips Fuel Co., 138 Northwestern, 547 (Iowa), November, 1912.

DANGERS IN ADJACENT WORKINGS.

A miner does not assume the risk of dangers in his place of work resulting from the condition of adjacent workings not under his control nor subject to his inspection.

Gennaux *v.* Northwestern Improvement Co., 130 Pacific, 495 (Washington), February, 1913.

FAILURE OF MINE OPERATOR TO FURNISH SAFE PLACE.

A miner by accepting employment in a coal mine assumes the risk of such accidents and resulting injuries as may happen to him in the ordinary and customary course of his employment, but he does not assume any risk from accident or injury caused by the failure of the mine operator to exercise ordinary care to furnish him a reasonably safe place in which to work, as the risks which he assumes does not embrace or include risks that are brought about by the failure of the master to perform his duty.

Fluehart Collieries Co. *v.* Elam, 151 Southwestern, 34 (Kentucky), December, 1912.

INJURY TO BOY OF PROHIBITED AGE.

A mining company employing a boy in its coal mine in violation of the Alabama statute can not, in an action by such boy for injuries resulting to him in the course of such employment, invoke the defense of assumption of risk.

De Soto Mining & Development Co. *v.* Hill, 60 Southern, 583 (Alabama), December, 1912.

INEXPERIENCED AND YOUTHFUL MINER.

A youthful and inexperienced miner does not assume the risk of injury from falling rock or slate where he was directed by the foreman to procure coal for an engine and where he blasted it as directed by the foreman, and where he was ignorant as to any danger from rock or slate falling from the roof and had not been advised as to such danger or instructed in any manner in respect to his duties, except as to how and where to shoot the coal.

Bartley *v.* Elkhorn Consolidated Coal & Coke Co., 152 Southwestern, 955 (Kentucky), January, 1913.

DEFECTIVE MACHINERY.

A miner does not and can not assume risks caused by the negligence of the mine operator, and although the breakage of a feed chain used to draw a coal-mining machine might occur on occasions of unusual and unavoidable stress on the machine, and accordingly the risks of injury from such causes being natural and incidental to the service would be assumed by a miner operating the machine, yet the miner did not assume the risk resulting from continuing to use a chain so defective as to be unsafe for ordinary uses.

Bradley *v.* Northern Central Coal Co., 151 Southwestern, 180 (Missouri), November, 1912.

DEFECTIVE APPLIANCES.

A miner was engaged with others in sinking a prospect shaft, and for the purpose of carrying on the work and removing the dirt excavated a car was operated over a track extending up to and over a part of the collar of the shaft, and a wooden block or bumper was placed over one of the rails to prevent the car from passing over the

end of the rail, and by reason of the car wheels coming in contact with the block it became worn and defective and the car passed over it and precipitated the miner into the shaft to his injury. In an action by the miner to recover for such injuries it was determined that the miner was not called upon to conduct a series of experiments to ascertain the extent of the damage occasioned by each contact of the car wheels with the block or the force necessary to move the car over the block, and unless the bumper was manifestly insufficient to perform the service required of it the miner is not chargeable, as a a matter of law, with the assumption of the risk.

Killeen *v.* Barnes-King Development Co., 127 Pacific, 89 (Montana), October, 1912.

DANGER FROM LIVE ELECTRIC WIRES.

A court can not decide as a matter of law that a miner assumed the risk of dangers arising from the presence of a live trolley wire where he knew that contact with such wire would shock a person, but did not know there was sufficient power to seriously hurt or harm a person.

Williams *v.* Bunker Hill & Sullivan Mining, etc., Co., 200 Fed., 211, October, 1912.

DANGER FROM UNGUARDED ELECTRIC WIRES.

The fact that a miner has knowledge of some danger by contact with a live electric wire does not necessarily imply that the results of such contact are appreciated by the miner, and it can not be said, as a matter of law, that a miner unskilled in the uses of electricity comprehends or appreciates that if, while obeying his superior and performing his ordinary work, he should touch an uncovered wire with a metal-bound hose he would receive probably severe or fatal injuries by reason of the electric current passing through his body; and though a miner may have knowledge of such danger without an actual appreciation of the risk to which he is exposed, and though the miner assumes the ordinary risks and dangers of working in a mine, as well as the ordinary risks which he knows and appreciates, yet he does not assume the risk of the employer's negligence where the effect of such negligence is not plainly to be observed by the miner upon the reasonable use of his senses.

Williams *v.* Bunker Hill Mining, etc., Co., 200 Fed., 211, October, 1912.

CHANGING CONDITIONS.

The rule that a miner assumes the risk where the conditions are constantly changing and the risks are incident to the operation does not apply where such changed conditions are in an adjacent chute or chamber and are not due to the progress of the work of the miner, but to a "squeeze" of which the operator had knowledge and of which the miner himself had neither knowledge nor warning.

Gennaux *v.* Northwestern Improvement Co., 130 Pacific, 495 (Washington), February, 1913.

FAILURE TO TIMBER ROOF.

An employee in a mine whose duties were those only of a loader does not assume the risk of the operator's failure to properly timber and support the roof of an entry in the mine, where such employee had nothing to do with respect to making the working place safe.

Peabody Alwert Coal Co. *v.* Yandell, 100 Northeastern, 758 (Indiana), February, 1913.

WANT OF KNOWLEDGE OF DANGEROUS STAIRWAY.

A mine employee who had made but one trip down into the shaft of a mine and was escorted down into the mine by two other miners, the purpose being to show him the way down the shaft and where his attention was not called to the fact that the bottom flight of the stairway was without banisters, did not on entering the mine on the second day to begin work assume the risk of an injury due to his falling on such lower stairway while reaching for the missing banister after the light on his cap had been extinguished by some falling substance.

Consumers Lignite Co. *v.* Hubner, 154 Southwestern, 249 (Texas), March, 1913.

CONVICT MINER NOT CHARGED WITH THE ASSUMPTION OF RISKS.

A convict working in a mine under involuntary servitude can not be held to have assumed the risk of negligence on the part of the coal operator's employees in the mine nor of his fellow convicts at work in the mine.

Sloss-Sheffield Steel & Iron Co. *v.* Weir, 60 Southern, 851 (Alabama), January, 1913.

I. PROMISE OF OPERATOR TO REPAIR—EFFECT.

MINER MAY RELY ON PROMISE.

A miner has the right to rely on the promise of a mine foreman to remedy the dangerous condition of the roof of an entry in a mine where the miner was carrying out the immediate directions and orders of such foreman and where the danger was not imminently obvious.

Broadway Coal Mining Co. *v.* Robinson, 150 Southwestern, 1000 (Kentucky), November, 1912.

A miner with but slight acquaintance with the particular mine in which he was working may rely on the promise of the mine operator to do the necessary timbering as soon as the timbers could be gotten into the mine from the surface, although he knew that the roof of his room sounded "loose and drummy," but the pit boss assured him that he knew from his greater experience the place was safe and ordered him to continue work, and where the danger was not so apparent that a reasonably cautious man would have seen and appreciated it.

Braddich *v.* Phillips Coal Co., 138 Northwestern, 406 (Iowa), November, 1912.

MINING LEASES.

A. LEASES GENERALLY—CONSTRUCTION
B. COAL LEASES.
C. OIL AND GAS LANDS.
D. PHOSPHATE LANDS.

A. LEASES GENERALLY—CONSTRUCTION.

LEASEHOLD INTEREST IN MINING CLAIM.

A leasehold interest in a mining claim under a lease for five years is not a freehold estate.

Equitable Mines Corporation *v.* Maxwell, 127 Pacific, 243 (Colorado), October, 1912.

DEVELOPMENT AND FORFEITURE.

Leases of lands for the exploration and development of minerals are generally executed by the lessor in the hope and upon the condition, express or implied, that the land shall be developed for minerals; and it would contravene the nature and spirit of such a lease to permit the lessee to hold it any considerable length of time without making any effort whatever to develop the land according to the express or implied purpose of the lease. And though equity abhors a forfeiture, yet when such a forfeiture works equity and is essential to public and private interests in the development of minerals in land, then the lessor as well as the public will be protected from the laches of a lessee, and a forfeiture under such circumstances will be decreed.

Killebrew *v.* Murray, 151 Southwestern, 662 (Kentucky), December, 1912.

SURRENDER OF LEASE—FORFEITURE.

A common form of lease of undeveloped lode mining property which the lessor seeks to have developed at the expense of the lessee who assumes the burden of expense of developing the property in the hope of finding a paying mine is nonforfeitable so long as the lessee properly performs the required labor, but failing to perform the required work the lessor may forfeit the lease, and the lessee may after the performance of a certain amount of labor upon the property abandon or surrender the lease, if the prospect for profitable mining will not warrant a continuation, as the law assumes that it was in the contemplation of the parties that the showing of conditions upon the leased premises that would not justify further expenditures upon the part of the lessee warrants him in terminating the lease.

Girton *v.* Daniels, 129 Pacific, 555, p. 557 (Nevada), February, 1913.

DUTY OF LESSEE TO PROTECT SURFACE.

The lessee of a mining lease must protect the surface of the ground in which a mine is located.

Gulf Pipe Co. v. Pawnee-Tulsa Petroleum Co., 127 Pacific, 252 (Oklahoma), October, 1912.

LEASE NOT ACCEPTED.

A mining company can not be held liable under the terms of a lease which it never in fact accepted or acted upon.

Stiritz v. Big Muddy Mining Co., 100 Northeastern, 968 (Illinois), February, 1913.

DELIVERY OF LEASE IN VIOLATION OF CONDITIONS.

A person in the possession of a mining lease and authorized to deliver the same to the lessee only after it had been submitted to and approved by a certain named person can not make a valid delivery of such lease to the lessee without its being submitted to and approved by the person named.

Stiritz v. Big Muddy Mining Co., 100 Northeastern, 968 (Illinois), February, 1913.

CONSIDERATION FOR ASSIGNMENT OF LEASE.

The assignment of a mining lease for the development of mining property, where work done under the lease may disclose a mine of great value, is a sufficient consideration to support an agreement on the part of one assignee to bear one-third of the expenses of developing the mine, though the subsequent development of the property shows that the lease was in fact valueless.

Girton v. Daniels, 129 Pacific, 555, p. 557 (Nevada), February, 1913.

ORAL AGREEMENT TO SHARE IN LEASE.

An oral agreement by which three persons agree to be equal coowners of a mining lease and each contributes the equal one-third part of the work and expenses in developing the mining claims covered by the lease, and share equally in any profits arising from the business, is not void as being within the statute of frauds, where the lease under its provisions could have been terminated by the act of the parties within a year, and one of the three persons failing to contribute his part of the assessment work is liable to the others.

Girton v. Daniels, 129 Pacific, 555 (Nevada), February, 1913.

B. COAL LEASES.

SUIT TO CANCEL COAL LEASE.

A lessor may maintain a bill to cancel and annul a coal-mining lease where the terms of the lease have been violated by reason of the failure to pay rents and royalties, and failure to work the mines with

reasonable diligence, and failure to open and operate the mine in a skillful and minerlike way, and where it is averred that the defendant, assignee of the lease, is insolvent and unable to work the property as contemplated by the lease.

Peerless Coal Co. *v.* Lamar, 60 Southern, 837 (Alabama), January, 1913.

RIGHTS UNDER COAL LEASES.

A lease by which the lessor leased certain coal lands, together with certain surface privileges for mining purposes, to the lessee for a stated period or until the coal and minerals were worked out does not authorize the lessee to use the leased premises or the openings and apertures to the mine on the leased lands for the purposes of mining coal from adjacent lands not belonging to the lessor, and such improper use of the leased lands by the lessee will be enjoined, as the uses expressly granted are equivalent to a negative covenant on the part of the lessee that the leased premises would not be used for purposes other than those stipulated.

Brasfield *v.* Burnwell Coal Co., 60 Southern, 382 (Alabama), December, 1912.

RIGHTS OF LESSEE UNDER LEASE FOR SEPARATE TRACTS OF COAL LANDS.

A lessor who leased a particular tract of coal land to the lessee for a stated period or until the coal or minerals were all worked out, with the necessary surface rights, may enjoin such lessee from using the surface rights for the mining of coal from an entirely separate tract of land held under another and distinct lease from a different lessor.

Brasfield *v.* Burnwell Coal Co., 60 Southern, 382 (Alabama), December, 1912.

RIGHT OF LESSEE UNDER COAL LEASE.

A lease of separate tracts of coal lands to a lessee for a stated period or until the coal was mined out, with the necessary surface privileges for mining, did not authorize the lessee, after mining the coal from one tract, to extend its tunnels from the tract so mined through adjoining coal lands owned by a different person in order to reach a separate tract held under his lease, and to mine out, by extending such tunnels and by the use of the surface privileges, the coal in such adjoining tract, though this was the most practical and economical method of mining under his lease the coal in the separate tract.

Brasfield *v.* Burnwell Coal Co., 60 Southern, 382 (Alabama), December, 1912.

AUTHORITY TO DUMP REFUSE ON SURFACE.

A lease of coal lands gave the lessee certain surface privileges for the use of mining, including the dumping of slate taken from the leased premises in the operation of the mine, but it did not either

expressly or impliedly authorize the lessee to dump slate on the surface taken from coal mines held under a separate and distinct lease from other lessors.

Banfield v. Burnwell Coal Co., 60 Southern, 382 (Alabama), December, 1912.

C. OIL AND GAS LEASE.

CONSTRUCTION.

Oil and gas leases must be construed in the light of surrounding circumstances. It is a matter of common knowledge that anyone can not estimate the exact time within which a well can be completed; that delays due to trivial or grave accidents and other causes beyond the possibility of accurate anticipation are likely to occur; and that under such circumstances adherence to the strict letter of an extension clause of a lease might inflict disastrous losses upon diligent and honest lessees, a consequence obviously not within the intent of either party. Such leases are drawn for all sorts of territory, some known to be rich in minerals and others not known to contain any, and accordingly their terms are varied to allow for such differences, and in territory remote from profitable operation such leases are often taken without reasonable expectation of immediate advantage to the lessor other than a nominal rental in the form of delay money and with the expectation of delay in drilling until neighboring lands are shown to contain minerals sufficient to justify exploration and operation.

South Penn Oil Co. v. Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

IMPLIED COVENANT.

There is always an implied covenant on the part of a lessee in an oil and gas lease to operate the leased property for the mutual benefit of both parties, and such covenant imposes on the lessee a duty to drill as many wells on the leased premises as are necessary to obtain therefrom all the oil and gas that is reasonably fair and profitable to extract and market; but this implied covenant is subject to the right of the lessees at any time wholly to cease operations and to abandon the property, in the absence of an express covenant to operate the premises, and under such implied covenant the lessee has an option to give up his lease at any time without liability, or to exhaust the mineral resources of the leased premises, but choosing the latter he must operate in a skillful, diligent, and just manner as long as he occupies and uses the premises.

Hall v. South Penn Oil Co., 76 Southeastern, 124 (West Virginia), October, 1912.

DILIGENCE AND GOOD FAITH—OPERATIONS AND DISCOVERY.

Under the ordinary oil and gas lease mere discovery of mineral vests an estate, and without actual production gives a right to continue operations, and cessation of actual production is excused by

adverse conditions and miscarriages which must have been contemplated by the parties, though not specified in the lease; the tests of duty and right are diligence and good faith in almost all cases when the terms read in the light of the conditions and circumstances will permit their observance.

South Penn Oil Co. *v.* Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

REASONABLE TIME TO ENTER AND PROSPECT.

While a "reasonable time" to enter upon premises and prospect under an oil and gas lease is in a sense uncertain, vague, and indefinite, and may be dependent upon surrounding circumstances, yet a lessee under such a lease acquires substantial rights that will receive the same consideration and protection at law and in equity as will those of a lessee whose period for entry and prospecting is definitely fixed.

Smith *v.* Guffy, 202 Federal, 106, October, 1912.

PRODUCTION OF OIL IN PAYING QUANTITIES.

A stipulation in an oil and gas lease to the effect that the term shall continue as much longer thereafter as oil or gas shall be found in paying quantities requires that oil or gas shall be actually discovered and produced in paying quantities within the term; and, under such stipulation, it is not enough to complete a well having some indications of oil, or a well that might be developed into a well producing oil in paying quantities, but the lessee must actually find oil in paying quantities.

South Penn Oil Co. *v.* Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

LESSEE ENTITLED TO EXCLUSIVE POSSESSION.

A lessee entitled to the exclusive right of gas and oil produced on the tract of land under an unexpired lease from the owner of the land may enjoin any trespass thereof, and the threatened drilling of a well by a stranger for the purpose of extracting the oil and gas, as the damages that would accrue in such a case are regarded as incapable of definite ascertainment and an action at law does not give adequate relief.

Campbell *v.* Smith, 101 Northeastern, 89 (Indiana), March, 1913.

EXCLUSIVE RIGHT—EFFECT OF RELEASE.

An oil and gas lease gave the lessee the exclusive right to drill and sink wells for oil and gas on the leased premises and to take and remove the same therefrom, and the right to dig and use water wells thereon, with the exclusive right to lay pipes and mains, and an ex-

clusive right of way to conduct the oil and gas from and through the leased premises. Subsequently, the lessor platted a portion of the leased premises as an addition to a village and dedicated streets and alleys to the public, and thereupon the lessee, for a valuable consideration, released all right to drill for oil, gas, or water on the lots or streets so platted, as well as the right to lay pipes and mains in or across such platted premises; and the release further provided that if the lessor thereafter granted the right to drill for oil or gas upon any of such lots or streets or to remove oil or gas therefrom or lay gas or oil pipes therein, then the original lease shall be reinstated in full force and effect. In an action between the lessee and a gas company supplying gas as a public-service corporation to the inhabitants of the village it was decided by the court that the grant of the right by the village to such public-service corporation to lay its pipes and mains through the streets of such platted part of the original leased premises did not, under the terms of such release, revive the exclusive right of the original lessee to carry on its operations upon such platted and released part of the original leased premises, as the original lessor had nothing to do with, and no power over, the granting of the right to such public-service company to lay its pipes and mains in the public streets, and where no right was given to such public-service company by the lessor to drill or sink any such enumerated wells upon any of the lots of such platted portion of the original leased premises.

Carroll *v.* Silver Creek Gas & Improvement Co., 139 New York Supp., 161, October, 1912.

LESSEE CAN NOT MAINTAIN INJUNCTION BEFORE PROSPECTING.

A lessee in an oil and gas lease providing for the payment of a royalty to the lessor on the production of oil or gas, and requiring the lessee to complete a well within a stated time or pay a stipulated sum during the period of delay, and giving the lessee the option to cancel it at any time on payment of a nominal sum, can not, prior to the discovery of any oil or gas by him, maintain a suit in equity against either the lessor or a subsequent lessee who has with notice of the prior lease entered upon and is operating the premises, for the protection and enforcement of the right conferred by the lease, but must depend solely upon his remedy at law.

Smith *v.* Guffy, 202 Fed., 106, October, 1912.

LESSEE MAY ENJOIN SUBSEQUENT LESSEE FROM OPERATING.

An oil and gas lease for certain described premises, with the right to enter thereon and with the ordinary privilege of operating the same for oil and gas, gives to the lessee a vested interest in the oil obtained from such premises and authorizes the lessee to maintain

an action for damages and to enjoin a subsequent lessee of the same premises claiming under a second lease from the landowner from operating the premises and from taking and removing oil therefrom.

Kahle *v.* Crown Oil Co., 100 Northeastern, 681 (Indiana), January, 1913.

SECOND LESSEE NOT WILLFUL TRESPASSER.

Where a court of general and competent jurisdiction declared an oil and gas lease invalid and canceled the same and on the faith of such determination the landowner leased the same premises to another person who thereupon operated the same and mined and removed oil therefrom, such second lessee can not be regarded as a willful trespasser and held liable in damages as such where the original lessee subsequently appealed his case to the supreme court and the judgment of the lower court was reversed and his lease held valid.

Kahle *v.* Crown Oil Co., 100 Northeastern, 681 (Indiana), January, 1913.

DUTY AS TO PLACE OF DRILLING.

The lessee of oil and gas under a large tract of land can not drill a well where it will endanger the property and lives of others who are lawfully using the surface when such lessee can drill at other places that are equally advantageous to him and will not endanger the lives and property of those using the surface.

Gulf Pipe Line *v.* Pawnee-Tulsa Petroleum Co., 127 Pacific, 252 (Oklahoma), November, 1912.

ASSIGNMENT OF LEASE—EFFECT AND VALIDITY.

Under the statute of Pennsylvania a prothonotary of a common pleas court is not authorized to take the probate of an unacknowledged assignment of an oil and gas lease to qualify it for record, and the record of such an assignment is a nullity and confers no right upon the assignee.

Midland Gas Co. *v.* Jefferson County Gas Co., 85 Atlantic, 853 (Pennsylvania), November, 1912.

ASSIGNMENT OF OIL LEASE AND RIGHTS OF ASSIGNEE.

A pretended assignee claiming an oil and gas lease under an invalid assignment can not estop the lessor from executing a second lease, the original having been surrendered and canceled, by inducing him to accept the rentals stipulated in the first lease for a failure to complete a well on the premises within the specified time by falsely representing that a fraud had been perpetrated in the cancelation and surrender of the lease and that he was in fact the rightful owner of such a lease.

Midland Gas Co. *v.* Jefferson County Gas Co., 85 Atlantic, 853 (Pennsylvania), November, 1912.

LEASE OF INDIAN LANDS—ASSIGNMENT.

The recordation laws of Arkansas extended to and put in force in the Indian Territory before statehood, and there construed, do not require the recording of an assignment of an oil and gas lease of Indian lands in order to give the assignment validity.

Scott *v.* Signal Oil Co., 128 Pacific, 694 (Oklahoma), December, 1912.

LEASE OF INDIAN LANDS—ASSIGNMENT OF ROYALTY.

An assignment of royalty due under a gas and oil lease given by a Quapaw Indian on his allotment of land is not an alienation of a part of his land within the meaning of the act of March 2, 1895 (28 Stat., 907), making such allotments inalienable for 25 years.

Wat-tah-noh-zhe *v.* Moore, 129 Pacific, 877 (Oklahoma), January, 1913.

LIABILITY OF ASSIGNEE.

In an action by a lessor of an oil and gas lease against an assignee of the lease, where the fact of the assignment of a lease arose out of a transaction, of which the lessor was not cognizant, between the lessee and the defendant in the action, proof of possession of the leased premises by such defendant is sufficient to warrant an inference that the lease is assigned and that the defendant is in possession of the premises as an assignee of the lessee, and it may be inferred that a contract between such assignee and the lessor, as to the drilling of additional wells, was made with reference to the original lease, and especially where it is shown that the assignee had in his possession the original lease.

Indiana Natural Gas & Oil Co. *v.* Duling, 100 Northeastern, 96 (Indiana), December, 1912.

FAILURE TO OPERATE—REMEDY OF LESSOR.

Under an oil lease the failure of the lessee to drill additional wells under conditions imposing the duty to do so makes the lessee liable in damages to the lessor for which the remedy at law is adequate, but such failure affords no ground for either part or entire cancellation or rescission of the lease, in the absence of a stipulation to that effect.

Hall *v.* South Penn Oil Co., 76 Southeastern, 124 (West Virginia), October, 1912.

TERMINATION AND EXTENSION OF LEASE.

Under an oil and gas lease for a term of 10 years, and as long thereafter as oil or gas is produced from the leased premises, the discovery of oil, though in unremunerative quantities, by the drilling of a well within the specified term of 10 years, will extend the term where there

has been a faithful, diligent, and expensive effort to make the well produce in paying quantities and otherwise develop the leased premises, under the rule that discovery of oil within the term vests an estate, terminates the mere right of exploration, and creates the relation of landlord and tenant until the end of the fixed term, and the estate so vested can be brought to an end within the term only by a declaration of forfeiture for abandonment or surrender.

South Penn Oil Co. *v.* Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

CONDITIONS AND FORFEITURE.

A lease of oil lands providing that if a well was not drilled on the premises within one year the lease should be void, unless the lessee should within 60 days from the beginning of each year pay a rental of 25 cents per acre until a well was drilled or the lease canceled, requires the lessee either to drill a well within one year or to pay the required rental, and is in effect an option agreement, and by the lessee failing to do either the lease became forfeited and no rent was due thereunder.

Deming Investment Co. *v.* Lanham, 130 Pacific, 260 (Oklahoma), February, 1913.

BREACH OF CONDITION—FORFEITURE.

An oil and gas lease requiring the lessee to commence operation within four months and continue with diligence until completed, unavoidable delays and accidents excepted, and providing that the lease shall, upon the failure to complete a well as provided, be null and void, is forfeited and is of no effect where the lessee drilled one unproductive well and drilled another well to a depth of about 1,200 feet and then abandoned the premises, removed all drilling machinery, and made no further effort to develop the leased premises; and such abandonment by the lessee was conclusive on the question of his construction of the lease, and conclusively shows that he understood that the lease became void by its express terms, and he could, therefore, confer no right by a subsequent pretended assignment of the lease.

Shannon *v.* Long, 60 Southern, 273 (Alabama), December, 1912.

CANCELLATION—PLEADING.

In an action to cancel an oil lease because of the lessee's failure properly to develop the premises and to drill wells sufficient to prevent loss by drainage of the oil or gas from the leased premises through wells owned by the lessee on adjacent lands the complaint must state facts showing with reasonable certainty such drainage in substantial quantities, and should show a clear intent on the part

of the lessee to obtain the oil and gas from the leased premises through his own wells on adjacent lands by his failure and refusal properly to develop and drill on the leased premises.

Hall *v.* South Penn Oil Co., 76 Southeastern, 124 (West Virginia), October, 1912.

CANCELLATION—WAIVER.

A vendor of real estate reserving a vendor's lien to secure the unpaid purchase money, and reserving also one-half the oil and gas can not, after the foreclosure of his lien and the purchase of the property on decretal sale, maintain an action to cancel an oil lease on the land executed by the vendee during his rightful possession where such vendor has continued to receive and accept the rentals payable for such one-half of the oil and gas reserved by him, as he thereby recognized the right of his vendee to make a lease of the entire oil and gas interest, on payment to him of such rentals for the oil and gas thus reserved.

Batten *v.* Hope Natural Gas Co., 76 Southeastern, 889 (West Virginia), December, 1912.

LEASE NOT AFFECTED BY SUBSEQUENT DEED.

A landowner leased his lands to a lessee for oil and gas development, the lease providing for the delivery to the lessor, in a pipe line, of a stipulated part of all oil produced and for the payment to the lessor of a stipulated sum per year for each gas well used. A subsequent deed by the landowner granting a certain portion of all the oil and gas found within a part of such leased tract did not grant such stipulated part of the oil and gas in place in the ground, but only passed to such grantee the right to have delivered in the pipe line, by the lessee, such stipulated part of the oil produced and the stipulated part of the rental for the gas wells.

Horner *v.* Philadelphia Co., etc., 76 Southeastern, 662 (West Virginia), November, 1912.

JOINT LESSORS—SINGLE LEASE.

An oil and gas lease executed by three several owners of contiguous lands that describes the leased lands as a single tract without any intimation of several ownerships of parts thereof, or any suggestion of intent to make separate tenancies, and stipulates for a single well on the premises, or in lieu thereof the payment of a lump sum of money as rental, and provides for the liquidation of all rentals upon the completion of the well, is, as between the lessors and the lessee, a joint lease of a single tract of land, and the drilling of a well and production from any place on the premises would be a compliance with the terms of the lease and would, according to its provisions, extend the lease for another term of equal duration.

South Penn Oil Co. *v.* Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

CONDITIONAL TENANCY.

An oil and gas lease for a stated term of years and as much longer as oil or gas is produced from the leased premises that recites a nominal consideration and imposes conditions for nonperformance by which forfeiture is imposed, but contains no covenant on the part of the lessee to drill a well or to pay money for failure to do so, creates only a conditional tenancy that until its expiration is binding upon the lessor in accordance with the terms of the lease, if the stipulated conditions are performed.

South Penn Oil Co. *v.* Snodgrass, 76 Southeastern, 961 (West Virginia), January, 1913.

CONTRACT FOR SELLING OIL LEASES—STATUTE OF FRAUDS.

A parol contract for the sale of oil leases is within the statute of frauds, but where such an agreement has been executed and the contract of sale fully performed by proper transfer of the leases and nothing remains to be done except paying the purchase price, the statute of frauds can not be invoked as a means of defeating an action for the purchase price.

Love *v.* Kirkbride Drilling & Oil Co., 129 Pacific, 858 (Oklahoma), January, 1913.

OWNERSHIP OF OIL IN PLACE.

An oil and gas lease of certain described real estate with the exclusive right of drilling for gas and oil does not create in the lessee an ownership of the oil and gas in the earth, but gives to the lessee the exclusive right to mine for oil and gas on the premises, and as oil and gas are not the subject of property until reduced to possession, the lease is no more than an agreement for the exclusive right to prospect and market the product, and creates the difference between a lease of mines and a license to work mines—the former being a distinct conveyance of an actual interest or estate in lands, while the latter is only the incorporeal right to be exercised in the lands of others and may be held apart from the possession of the land.

Campbell *v.* Smith, 101 Northeastern, 89 (Indiana), March, 1913.

D. PHOSPHATE LANDS.

REASONABLE TIME FOR BEGINNING OPERATION.

In the absence of a provision as to the time for the commencement of operations in a lease of lands for mining phosphate, the presumption is that operations were to begin within a reasonable time; but where the lease fixed a time within which operations should begin, then the time fixed must be regarded as a reasonable

time, and on failure to begin the mining operations within the time fixed, or within a reasonable time in the absence of any fixed time, such failure may be treated by the grantor as an abandonment by the grantee of the leased premises.

Killebrew *v.* Murray, 151 Southwestern, 662 (Kentucky), December, 1912.

TERMINATION OF LEASE.

A lease for mining phosphate from certain lands for a period of 10 years, and by the terms of which the lessee may at any time, by written notice, terminate the lease, is in legal effect terminable at the will of the lessor, under the rule that a lease terminable at the pleasure of the lessee is also terminable at the will of the lessor.

Killebrew *v.* Murray, 151 Southwestern, 662 (Kentucky), December, 1912.

ABANDONMENT.

The failure to begin mining operations within a reasonable time under a lease of phosphate lands may be construed as an abandonment of the lease, the same as the failure to begin operation under an oil or gas lease; but the failure of a lessee to proceed with the work of development would operate as an abandonment of an oil or gas lease in a shorter time than if such failure were to result in the case of a coal or phosphate lease, because of the greater danger of loss by delay in the former by reason of the migratory character of oil and gas.

Killebrew *v.* Murray, 151 Southwestern, 662 (Kentucky), December, 1912.

CANCELLATION OF LEASE.

A lease of certain land for the sole purpose of mining and excavating for phosphate and phosphate-bearing rock for a term of 10 years and so long thereafter as such phosphate or phosphate-bearing rock may be found in quantities considered by the lessee as profitable, and providing for a certain stipulated royalty per ton and the payment of an annual nominal sum whether the lands are mined or not, and providing that the lessee might determine when and how much of such land shall be mined during each year, evidences an intention on the part of a lessor to obtain an income or profit in royalties from the lessee's mining operation, and especially where it appeared that the lessee represented that the mining of the phosphate on such land would pay in royalties as much as $500 per acre, and where it also appeared that the lessee represented that he would begin the work of mining within a year or 18 months. It is not to be presumed in such case that the lessor would have encumbered his land with such a lease for the annual payment of a nominal amount, but it may be

presumed that the real consideration or inducement for the granting of the lease was the mining of the phosphate which would be commenced within a reasonable time. The lease is therefore lacking in mutuality and is only a unilateral executory contract, unenforceable by either party, and on failure of the lessee to begin mining operations within a reasonable time the lessor may compel a cancellation of the lease.

Killebrew *v.* Murray, 151 Southwestern, 662 (Kentucky), December, 1912.

MINING PROPERTIES.

A. TAXATION.
B. INSPECTION.
C. TRESPASS.

A. TAXATION.

TAXATION OF MINERALS.

The constitution and statutes contemplate that taxes shall be levied on all lands in proportion to the true value, and this end can not be attained by permitting mineral lands to escape the burden of all values except agricultural ones.

Board, etc., of Greene County *v.* Lattas Creek Coal Co., 100 Northeastern, 561 (Indiana), January, 1913.

TAXATION OF COAL IN PLACE.

Where there has been no severance of title the owner of the fee holds the land taxable for its entire value, including that of the underlying minerals; but where there has been a severance the owner of the fee or the surface is properly assessable with the value of the surface, and the owner of the mineral with its value, and on this theory coal in place is subject to taxation where there has been a severance from the surface ownership.

Board, etc., of Greene County *v.* Lattas Creek Coal Co., 100 Northeastern, 561 (Indiana), January, 1913.

TAXATION—RELEASE BY STATE.

The constitutional provision by which the State of Texas released to the owner or owners of the soil all mines and minerals thereon subject to taxation was curative in its nature and retrospective in its effect, and was intended as an extinguishment of the rights of the State in only those mines and minerals in soil owned at the time of its adoption, and accordingly the title of the State to all other mines and minerals in lands of the public domain remained unimpaired and unaffected. And the State could provide by statute that any

sale or disposition of mineral lands should be understood to be with the reservation of the minerals thereon, and to be subject to location as provided.

Cox *v.* Robinson, 150 Southwestern, 1149 (Texas), November, 1912.

ASSESSMENT OF MINERALS—METHODS OF VALUATION.

The difficulty in obtaining a correct valuation of coal in place does not of itself render such coal nontaxable, and under the statutes of Indiana the assessor in obtaining the value of coal in place has a right to examine the owner or the officers of a corporation in order to elicit from them the result of any exploration or developments for coal or mineral and to ascertain the purchase price of such coal and to consider any development in the immediate neighborhood as well as sales of mineral rights or other facts constituting elements of value.

Board, etc., of Greene County *v.* Lattas Creek Coal Co., 100 Northeastern, 561 (Indiana), January, 1913.

TAXATION OF MINERALS.

It is the duty of taxing officers to assess the entire real estate holdings of the owner not severed or detached by his own act as a single body, and they have no authority to assess them severally for the purpose of taxation; and this rule applies where a tract of land has been assessed as a whole and part of the surface or minerals has been sold.

McCormick *v.* Berkley, 86 Atlantic, 97 (Pennsylvania), January, 1913.

SEVERANCE OF SURFACE AND MINERALS.

Where a landowner sold an entire tract of land, including the mineral therein, and reserved a part of the surface only, the part so reserved was severed from the residue of the tract, but the whole of the residue, surface as well as mineral, continued to be a single body of real estate, the title to which was vested by the single conveyance in a subsequent owner, and it was the duty of the taxing officers to assess the surface and the minerals together as an entire body of real estate belonging to the owner, and the minerals should not be separated from the surface and assessed separately.

McCormick *v.* Berkley, 86 Atlantic, 97 (Pennsylvania), January, 1913.

PATENTED CLAIMS.

The constitution of Nevada provides for a uniform rate of assessment and taxation of all property, except mines and mining claims, and provides also that the proceeds of these, when not patented,

shall be assessed and taxed, and when patented each patented mine shall be assessed at not less than $500, except when $100 in labor has been actually performed on any such patented mine during the year in addition to the tax on the net proceeds. This is construed to mean that patented mines shall be assessed at not less than $500 if $100 in labor has not been actually performed upon such patented mine during the year, but if such labor is so performed upon a patented mine it becomes exempt from assessment and taxation, except on the net proceeds, the same as an unpatented claim. The evident purpose of the legislature was to stimulate the prospecting and operation of patented mines and not encourage the owners to have such mines lie dormant and unprospected, and to require them to pay a tax if they did not perform annual labor worth $100, but to exempt them from this tax if they did perform the labor.

Goldfield Consolidated Co. *v.* State, 127 Pacific, 77 (Nevada), October, 1912.

B. INSPECTION.

RIGHT OF STOCKHOLDER.

A stockholder of a mining corporation has the right to visit and inspect the mines of the corporation, and this right rests upon the theory that the persons in charge of the corporation are merely the agents of the stockholders who are the real owners of the property, and for the further reason that from the nature of mining property an inspection of the books would afford no information of the nature of the ore bodies exposed or the manner in which the work was carried on.

Hobbs *v.* Tom Reed Gold Mining Co., 129 Pacific, 781 (California), February, 1913.

C. TRESPASS.

MEASURE OF DAMAGES.

A person who, in the honest belief that he is in the lawful exercise of his rights, unintentionally enters upon the property of another and removes ore or minerals is liable in damages for the value of such ore or minerals in its original place, and for no more, but in case of a willful and intentional trespass the wrongdoer is liable for the full value of such ore or mineral taken at the time of its conversion without any deduction for the labor bestowed or expense incurred in mining the same.

Kahle *v.* Crown Oil Co., 100 Northeastern, 681 (Indiana), January, 1913.

WILLFUL TRESPASS—MEASURE OF DAMAGES.

In case of a willful or negligent trespass in mining and removing oil the trespasser is charged with the value of the oil taken, without any deduction for the cost of taking it out; but if the trespasser was inno-

cent of the fact that he was trespassing, and guilty of no negligence in entering upon the ground and extracting the oil, then the cost of extracting should be allowed him.

Campbell *v.* Smith, 101 Northeastern, 89 (Indiana), March, 1913.

TRESPASS ON VEIN BELOW SURFACE—REMEDY.

In an action by one mining company against another for damages, to determine the ownership of a vein having its apex within the exterior boundaries of a claim of the suing company and which extends on its dip outside of the exterior boundaries of such claim and underneath the surface boundaries of the defendant company's claim, and from which the defendant company is working and extracting large amounts of ore, the refusal of the trial court to grant an injunction to prevent the defendant company from working such vein and from removing the ore therefrom until the controversy is settled will not be disturbed on appeal unless it clearly appears that the trial court has abused its discretion.

Stewart Mining Co. *v.* Ontario Mining Co., 129 Pacific, 932 (Idaho), January, 1913.

DAMAGES FOR INJURIES TO MINERS.

ELEMENTS OF DAMAGES.

In determining the damages to be assessed for the death of a miner a jury may consider his age at the time of his death; his habits or industry, his ability to labor, his capacity to earn money, the wages earned at the time of his death, the amount he had been in the habit of contributing to his family, his expectancy, his possible illness, his possible want of employment, the effect upon his earning capacity of his advancing years, as well as the expectation of the lives of the members of his family.

Meola *v.* Quincy Mining Co., 140 Northwestern, 460 (Michigan), March, 1913.

A driver of coal cars injured by the falling of loose coal, through the negligence of the coal-mine operator, had a right of action against the operator, as soon as the injury occurred, for all damages he sustained, and the true measure for the loss of earning power is the present value of such damages during the expectancy of his life had the injury not occurred, and the fact that he was rendered a helpless and hopeless paralytic, with a total loss of earning power for the full period of his expectancy, was one element of his damages, and his bodily pain and mental anguish on account of his condition, which he must endure as long as he lives, are other and proper elements of damages.

Prairie Creek Coal Mining Co. *v.* Kittrell, 153 Southwestern, 59 (Arkansas), December, 1912.

PERMANENT INJURIES—LIFE TABLES AS EVIDENCE.

In an action by a miner for damages for permanent injuries, standard life tables showing the ordinary expectancy of life are admissible in evidence on the question of damages, the same as in actions for death, as in either case the question is one of the destroyed capacity to earn money, and whether this is a part or a total destruction is not material.

Proctor Coal Co. *v.* Beaver, 152 Southwestern, 965 (Kentucky), January, 1913.

EXCESSIVE DAMAGES.

In an action for the wrongful death of a miner 35 years of age and capable of earning $65 per month a judgment for $18,000 was regarded as excessive, and no recovery in excess of $14,000 should be permitted.

Delaski *v.* Northwestern Improvement Co., 127 Pacific, 421 (Washington), September, 1912.

A verdict for $5,040 for the death of a miner was regarded as excessive where, based upon the evidence of his past contributions to his family, the miner if he had lived and had kept up such contributions during his entire expectancy would have contributed less than $1,300.

Meola *v.* Quincy Mining Co., 140 Northwestern, 460 (Michigan), March, 1913.

DAMAGES NOT EXCESSIVE.

A verdict for $500 for damages for injuries to a miner who was mashed and bruised by a fall of slate and was laid up for several weeks and was compelled for a time to go on crutches and suffered from an abscess on his back can not be regarded as excessive.

Ada Coal Co. *v.* Linville, 153 Southwestern, 21 (Kentucky), February, 1913.

A judgment for $2,500 was not excessive for an injury to a miner where it appeared that his leg was broken and his ankle dislocated, and that after the fracture healed it left him with a permanently crooked and deformed leg.

Stony Fork Coal Co. *v.* Lingar, 153 Southwestern, 6 (Kentucky), February, 1913.

A court can not say as a matter of law that a verdict of $18,000 is excessive where a miner 29 years old, strong and well and capable of earning on an average of $4 a day, is crippled for life and can earn practically nothing, is partly paralyzed, and suffers constant pain.

Gennaux *v.* Northwestern Improvement Co., 130 Pacific, 495 (Washington), February, 1913.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 15. Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls., 5 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 51. The analysis of black powder and dynamite, by W. O. Snelling and C. G. Storm. 1913. 80 pp., 5 pls., 5 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust tests in the experimental mine near Bruceton, Pa., by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 5 pls., 28 figs.

BULLETIN 59. Investigations of detonators and electric detonators, by Clarence Hall and S. P. Howell. 1913. 79 pp., 10 pls., 5 figs.

BULLETIN 62. National mine-rescue and first-aid conference, Pittsburgh, Pa., September 23–26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 65. Oil and gas wells through workable coal beds; papers and discussions, by G. S. Rice, O. P. Hood, and others. 1913. 101 pp., 1 pl., 11 figs.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigation of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas analysis at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 46. Quarry accidents in the calendar year 1911, by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 52. Permissible explosives tested prior to March 1, 1913, by Clarence Hall. 1913. 11 pp.

MINERS' CIRCULAR 3. Coal-dust explosions, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp., 5 figs.

MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 9. Accidents from falls of roof and coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.

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Bulletin 62

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

NATIONAL MINE-RESCUE AND FIRST-AID CONFERENCE

PITTSBURGH, PA., SEPTEMBER 23–26, 1912

BY

HERBERT M. WILSON

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NATIONAL MINE-RESCUE AND FIRST-AID CONFERENCE, PITTSBURGH, PA., SEPTEMBER 23-28, 1912.

By HERBERT M. WILSON.

INTRODUCTION.

The act (36 Stat., 369) that established the Bureau of Mines in the Department of the Interior defined as part of the bureau's province and duty the making of "diligent investigation of the methods of mining, especially in relation to safety of miners and the appliances best adapted to prevent accidents," and it authorized the bureau from time to time to make such public reports of its work and investigations as the Secretary of the Department of the Interior may direct, with the recommendations of the bureau.

The national mine-rescue and first-aid conference was projected in the hope that it would be a factor in increasing safety in the mining industry through the recommendation of better and safer methods of conducting mine-rescue operations, more efficient methods of extending and giving training in the use of artificial-breathing apparatus, and approved and more effective methods of administering first aid to injured miners and of training miners in first-aid work.

The bureau feels warranted in publishing the resolutions adopted by the conference as a guide to mine workers and others concerned in obtaining greater safety in mining, and in presenting such extracts from the discussion as indicate the reason for the adoption of any particular method. In issuing this bulletin, however, the bureau does not indorse these resolutions in whole or in part as a final expression of its views regarding the best method or procedure in any case. It does believe, however, that the resolutions furnish an excellent starting point from which to develop a standard code of procedure in mine-rescue and first-aid work.

It will be noted that in some important cases further information is necessary before any standard procedure can be recommended, particularly as regards types of bandages, methods of resuscitation, the use of stimulants, the cumulative effect of poisonous gases, the diet of men engaged in rescue work, the physiologic effect of the pressure

of breathing apparatus on the head, the amount of oxygen required in training, and the effect of concentrated oxygen, high temperature, etc. Regarding these subjects the bureau, in cooperation with a subcommittee appointed by the conference, will make tests and obtain further information with a view to perfecting apparatus and methods.

It is contemplated that additional conferences of a similar nature may be held, probably coincident with the formation of a national organization or society, in which event the Bureau of Mines may issue from time to time technical papers or miners' circulars giving such of the results as the bureau may wish to recommend for adoption.

At the conference here reported there were present 11 surgeons, 23 mine operators, 2 State inspectors, 14 company inspectors or representatives of safety departments, and 14 employees of the Bureau of Mines interested in that branch of its work having to do with increasing safety in the mining industry. The earnestness and interest of those assembled evidenced that they felt that there was much to be done to place organized mine-safety efforts on a more uniform basis. These men devoted themselves diligently to three days of serious consideration of the problems before them.

In this report endeavor has been made to eliminate all unnecessary remarks; but in order that those who hold diverging views regarding the subjects discussed may have before them the arguments that led the conferees to vote for the 36 resolutions adopted, the discussion had in the committees and in open session is published practically in full. Moreover, for the information of those mining officials, operators, and surgeons who are unfamiliar with approved methods of first-aid training and training with breathing apparatus and safety devices and methods of rescue, it seems equally desirable that the few opening addresses and closing remarks made in general session should be published as an expression of the reasons for such work and for the conference.

An outcome of the deliberations of the conference was the appointment of a committee charged with organizing a permanent association for the development of safety in mines. It is believed that meetings similar to the conference will be held by that association.

EVENTS LEADING TO THE CONFERENCE.

The national mine-rescue and first-aid conference here reported was held in Pittsburgh, Pa., September 23–26, 1912, as a logical sequence to the first national mine-safety demonstration held in the same city the previous year.

In the year that had elapsed the spread of instruction and training in first-aid treatment of injured miners and in the use of artificial breathing apparatus and other appliances for mine rescue and fire-fighting work had penetrated to every coal field and to a number of

the metal-mining fields in the United States. Numerous public demonstrations and first-aid contests had been held in the territory extending from the coal fields of eastern Pennsylvania to those of Washington and from Alabama to Minnesota. Many mining companies had established safety departments, the officers in charge having as their chief duty the organization of mine-rescue and first-aid training and the introduction of better safety measures in mining.

There had developed as a natural consequence a desire for a general conference for the coordination and standardization of this work. Mine surgeons and others engaged in first-aid instruction taught different and in some cases conflicting methods of treating certain injuries. Varying standards as to the kind and amount of training necessary with artificial breathing apparatus were adopted by the various advocates. Various rules for the holding of first-aid contests and differing methods of rating and judging in these had rendered it difficult to compare the relative merits of the same kind of work in different coal fields.

The Director of the Bureau of Mines, at the suggestion of a number of mine officials interested in these matters, and with a view to standardizing and improving the character of this work as demonstrated by the bureau, authorized the assembling of a conference, the personnel of which should include representatives of the mine-operating and the mine-safety departments and mine surgeons from those mining companies that maintained organized safety training and instruction.

INVITATIONS ISSUED.

On August 5, 1912, invitations for the conference were sent to all such mine operators as were known to maintain first-aid and rescue corps and as were located within a reasonable distance of Pittsburgh. In addition, invitations were sent to a few of the more distant operators who had shown interest and sympathy in the work by attendance or representation at the first national mine-safety demonstration of the previous year.

The 25 mining companies enumerated below responded to the invitation by promising to be represented by mine officials and surgeons. It will be noted that the companies are well distributed throughout the country.

Cleveland Cliffs Iron Co., Ishpeming, Mich.; Colorado Fuel & Iron Co., Pueblo, Colo.; Continental Coal Corporation, Wallsend, Ky.; H. C. Frick Coke Co., Scottdale, Pa.; Hailey-Ola Coal Co., Haileyville, Okla.; Keystone Coal and Coke Co., Greensburg, Pa.; Knoxville Iron Co., Knoxville, Tenn.; Lehigh Coal and Navigation Co.. Lansford, Pa.; Lehigh Valley Coal Co., Wilkes-Barre, Pa.; Lehigh and Wilkes-Barre Coal Co., Wilkes-Barre, Pa.; Oliver Iron Mining Co., Duluth, Minn.; Oliver & Snyder Steel Co., Oliver, Pa.; Pardee Brothers & Co., Lattimer Mines, Pa.; Penn-Mary Coal Co., Hellwood, Pa.; Philadelphia &

Reading Coal and Iron Co., Pottsville, Pa.; Pittsburgh Coal Co., Pittsburgh, Pa.; Republic Iron and Steel Co., Youngstown, Ohio; Superior Coal Co., Gillespie, Ill.; T. C. Tipper & Co., Pittsburgh, Pa.; Tennessee Coal, Iron and Railroad Co., Birmingham, Ala.; Vandalia Coal Co., Terre Haute, Ind.

EXHIBITS SOLICITED.

On September 3 manufacturers of first-aid materials and mine rescue appliances were invited to exhibit their products at the conference, and the near-by manufacturers of explosives were asked to exhibit dummy cartridges of permissible explosives, also detonators, etc.

In response to this invitation exhibits of the apparatus indicated were displayed during the conference by the following manufacturers:

Fidelity International Agency, safety lamps; Hughes Bros., safety lamps; Edward Schenk, breathing apparatus; Draeger Oxygen Apparatus Co., breathing apparatus and pulmotor; Siebe, Gorman & Co., breathing apparatus; Westphalia Engineering Co., breathing apparatus; Johnson & Johnson, first-aid materials; American National Red Cross, first-aid materials; Burroughs, Wellcome & Co., first-aid materials; Western Electric Co., mine telephone; Portable Electric Safety Lamp Co., electric lamps; Servus Rescue Equipment Co., breathing apparatus.

DELEGATES PRESENT.

The following persons were present and participated in the proceedings of the conference. Their names are grouped alphabetically by occupation and employment:

MINE OFFICIALS.

G. W. Barager, general manager, Pardee Bros. & Co., Lattimer Mines, Pa.; H. P. Dowler, general superintendent, Penn-Mary Coal Co., Hellwood, Pa.; Edward Elliot, general manager, Hailey-Ola Coal Co., McAlester, Okla.; Thomas G. Evans, stable boss, Lehigh Valley Coal Co., West Pittston, Pa.; J. R. Fleming, mining engineer, Penn Central Power Co., Altoona, Pa.; A. Gluding, engineer, Lehigh Valley Coal Co., Mahanoy City, Pa.; G. W. Hutchinson, chief engineer, Keystone Coal and Coke Co., Greensburg, Pa.; J. E. Jones, superintendent, Martin Coke Works, Republic Iron and Steel Co., Martin, Pa.; William Johnson, general manager, Saline County Coal Co., Canton, Ill.; William Lewis, mine foreman, Hailey-Ola Coal Co., Haileyville, Okla.; P. F. Lynch, superintendent, Cross Mountain Mines, Knoxville Iron Co., Briceville, Tenn.; J. M. McHugh, Tennessee Coal, Iron and Railroad Co., Ensley, Ala.; M. C. McHugh, Tennessee Coal, Iron and Railroad Co., Birmingham, Ala.; B. J. Matteson, assistant manager, Colorado Fuel and Iron Co., Pueblo, Colo.; J. F. Meagher, general superintendent of coal mines, Tennessee Coal, Iron and Railroad Co., 917 Amery Street, Pratt City, Ala.; R. B. Moss, superintendent of mines, Continental Coal Corporation, Arjay, Ky.; Charles Mountain, mine foreman, Crabtree, Pa.; F. B. Nold, general superintendent northern coal mines, Republic Iron and Steel Co., Republic, Pa.; J. P. Reese, general superintendent, Superior Coal Co., Gillespie, Ill., Consolidation Coal Co., Buxton, Iowa; T. I. Stephenson, president and manager Cross Mountain Mines, Knoxville Iron Co.,

Knoxville, Tenn.; J. D. Thomas, superintendent, Republic Iron and Steel Co., Gans, Pa.; T. C. Tipper, general manager, T. C. Tipper & Co., Jenkins Arcade Building, Pittsburgh, Pa.; H. P. Zeller, superintendent, Republic Iron and Steel Co., Republic, Pa.

MINE SURGEONS.

G. H. Halberstadt, Philadelphia & Reading Coal and Iron Co., Pottsville, Pa.; J. W. Kennihan, Pittsburgh Coal Co., Sharpsburg, Pa.; A. F. Knoefel, Vandalia Coal Co., Linton, Ind.; D. H. Lake, Delaware, Lackawanna & Western R. R., Scranton, Pa.; R. F. McHenry, Penn-Mary Coal Co., Hellwood, Pa.; F. L. McKee, Lehigh & Wilkes-Barre Coal Co., Wilkes-Barre, Pa.; J. W. Parshall, H. C. Frick Coke Co., Uniontown, Pa.; W. D. Richards, Knoxville Iron Co., Briceville, Tenn.; W. S. Rountree, Tennessee Coal, Iron and Railroad Co., Birmingham, Ala.; M. J. Shields, American National Red Cross, Washington, D. C.; J. H. Young, Lehigh Coal and Navigation Co., Lansford, Pa.

MINE-SAFETY OFFICIALS.

C. L. Albright, manager relief department, H. C. Frick Coke Co., Scottdale, Pa.; Atherton Bowen, assistant inspector of equipment, Lehigh Valley Coal Co., Wilkes-Barre, Pa.; Clyde G. Brehm, first-aid instructor, Oliver & Snyder Steel Co., Oliver, Fayette County, Pa.; William Conibear, safety inspector, Cleveland-Cliffs Iron Co., Ishpeming, Mich.; George H. Hawes, mine-rescue engineer, Oliver Iron Mining Co., Duluth, Minn.; Edward W. Judd, adjuster of claims, relief department, Pittsburgh Coal Co., Pittsburgh, Pa.; Austin King, chief inspector of mines, H. C. Frick Coke Co., Scottdale, Pa.; William Nisbet, inspector, Keystone Coal and Coke Co., Greensburg, Pa.; J. E. McDonald, superintendent relief department, Pittsburgh Coal Co., Oliver Building, Pittsburgh, Pa.; Henry R. Owens, engineer in charge of rescue station, Lehigh and Wilkes-Barre Coal Co., 76 Churst Street, Wilkes-Barre, Pa.; John L. Simons, fire inspector and first-aid instructor, Lehigh Coal and Navigation Co., Lansford, Pa.

STATE INSPECTORS.

J. F. Bell, Dravosburg, Pa.; F. W. Cunningham, Charleroi, Pa.

MISCELLANEOUS.

H. N. Elmer, general agent for Mexico and North America, Siebe, Gorman & Co. (Ltd.), 1140 Monadnock Building, Chicago, Ill.; T. B. Dilts, Y. M. C. A., Greensburg, Pa.; W. D. Roberts, mining instructor, public schools, Pittsburgh, Pa.

EMPLOYEES OF THE BUREAU OF MINES.

G. H. Deike, assistant mining engineer; Charles Enzian, coal-mining engineer; Jesse Henson, foreman miner; J. W. Paul, mining engineer; D. J. Price, foreman miner; G. S. Rice, chief mining engineer; C. O. Roberts, first-aid miner; J. C. Roberts, mining engineer; J. J. Rutledge, mining engineer; J. T. Ryan, assistant mining engineer; H. I. Smith, assistant mining engineer; C. S. Stevenson, foreman miner; E. B. Sutton, foreman miner; H. M. Wilson, engineer in charge of Pittsburgh experiment station.

DETAILED PROGRAM.

In pursuance of instructions by the director of the bureau, the author, as engineer in charge of the Pittsburgh experiment station,

in conference with J. W. Paul, engineer in charge of the mine-rescue and first-aid work of the bureau, and with the assistance of various mine surgeons and operators, and of employees of the bureau, arranged a program. Among those whose suggestions were most helpful were Drs. W. S. Rountree, A. F. Knoefel, M. W. Glasgow, R. F. McHenry, and J. W. Kennihan; mine operators T. I. Stephenson and B. J. Matteson; and Bureau of Mines foreman miners W. A. Raudenbush and C. O. Roberts.

The program outlined was as follows:

BUSINESS PROCEDURE.

MONDAY MORNING, SEPTEMBER 23.

9.30.—Opening address of welcome by Director Joseph A. Holmes.

Organization of conference.—H. M. Wilson, engineer in charge, Pittsburgh experiment station, Bureau of Mines, temporary chairman; C. S. Stevenson, Pittsburg, Kans., temporary secretary.

10.00.—Address by J. W. Paul: "Mine Rescue and First-Aid Work in the Bureau of Mines."

Informal addresses on mine-rescue and first-aid work in the various coal-mining regions.

11.30.—Appointment of committees; committees to be called together by their chairmen at 1.30 p. m. to-day.

12.00.—Adjournment until 9.00 a. m. to-morrow, when committee reports will be received.

TOPICS FOR DISCUSSION.

A. FIRST-AID METHODS.

1. Resuscitation from gas, electricity, and shock; methods of artificial respiration.
2. Treatment of dislocated shoulder and hip.
3. Splinting a broken back.
4. Applying splints to the limbs; should a long splint be used in all cases?
5. Should a first-aid man try to reduce a dislocation?
6. Should the triangular bandage be used in preference to the roller bandage?
7. Should a first-aid man attempt to wash a wound?
8. In case of a severe hemorrhage, the hemorrhage being controlled with a tourniquet, and no sterile dressing to be had, what is the proper thing to do?
9. Methods for transporting the injured; which is proper in various injuries?
10. Methods of rescuing patients from electric contact, and the most important thing to do first.
11. Method of conveying patient from place of injury to top of mine.
12. Development and acceptance of new methods and dressings.

B. FIRST-AID TRAINING.

1. What are the duties of a first-aid miner?
2. Instruction; its kind and amount.
3. Training and practice; kind and amount.
4. Systems of training: Red Cross, Navy, or new combination; value of a standard system for miners.

5. Methods of organizing first-aid corps.
6. Number of trained men per 100 miners.

C. FIRST-AID CONTESTS AND EXHIBITIONS.

1. Relative merits of contests and exhibitions; intracompany, intercompany, State, and interstate.
2. Standardization of methods of judging, number of judges, method of marking, etc.
3. Proper rating for speed.

D. MINE HOSPITALS.

1. Economic value to company and men of first-aid training and of mine hospitals.
2. Number, kind, and management of mine hospitals.
3. Underground first-aid stations and equipment.

E. BREATHING APPARATUS, SAFETY LAMPS, AND SAFETY APPLIANCES.

1. Methods of testing breathing apparatus.
2. Spare parts and repairs.
3. Safety and electric lamps.
4. Emergency equipment at rescue stations.
5. Use of birds and mice.

F. TRAINING WITH BREATHING APPARATUS.

1. Physical examination and requirements of persons given training.
2. Physiological effect of pressure on the head due to wearing helmet; physiological effect of pressure of nose clip.
3. Outline of course of instruction and training necessary to qualify for rescue work.
4. Diet of men doing rescue work.
5. Organization of rescue crews; how best to obtain volunteers for the work, compensation of volunteers, etc.
6. Amount of oxygen required in training.
7. Merits of smoke room and of outdoor and underground training.

G. CONDUCT OF RESCUE OPERATIONS.

1. Use of untrained men in rescue work.
2. Outline of procedure before and after entering a mine following explosions or mine fires.
3. Maximum distance rescue crews should proceed beyond fresh air.
4. Rest necessary for rescue men and limit of hours of work.
5. Cumulative effect of imbibing poisonous gases.
6. Use of stimulants.

CONFERENCE COMMITTEES.

At the opening session of the conference the chairman was authorized to appoint seven committees to consider the phases of the subject of the conference as outlined in the program. He was also authorized to appoint an eighth committee on permanent organization. The resolution directed that as secretary of each committee the chairman should designate an employee of the Bureau of Mines versed in

the subject matter to be considered by the committee, and that each committee should elect its own chairman.

The personnel of these committees, as finally organized, was as follows:

Committee 1—Mine-rescue apparatus and mine-rescue training.—J. W. Paul, chairman; D. J. Price, secretary; William Conibear, H. P. Dowler, Edward Elliott, D. K. Glover, William Nesbit, G. H. Hawes, Morgan Price, Austin King.

Committee 2—Rescue operations.—J. P. Reese, chairman; George H. Deike, secretary; B. J. Matteson, J. B. Meagher, H. P. Zeller, P. F. Lynch, F. W. Cunningham, William Raudenbush, William Johnson, W. J. German.

Committee 3—Safety lamps and electric lamps.—Austin King, chairman; J. T. Ryan, secretary; H. H. Clark, J. P. Bell, N. M. Lewis, J. M. McHugh, E. N. Judd, H. I. Smith.

Committee 4—First-aid methods.—W. S. Rountree, chairman; E. B. Sutton, secretary; Dr. W. G. Richards, Dr. A. F. Knoefel, Dr. J. W. Parshall, Dr. M. J. Shields, Dr. J. H. Young, Dr. J. W. Kennihan, Ralph McHenry, William Raudenbush, W. D. Roberts, Atherton Bowen, J. E. McDonald, Dr. G. H. Halberstadt, Dr. F. L. McKee.

Committee 5—First-aid training.—Dr. A. F. Knoefel, chairman; J. J. Rutledge, secretary; C. G. Bragh, Dr. M. J. Shields, Dr. J. W. Kennihan, Dr. F. L. McKee, Dr. J. H. Young, Dr. G. H. Halberstadt, Jesse Henson, E. W. Judd, T. B. Dilts, Dr. Ralph McHenry, J. E. McDonald.

Committee 6—First-aid contests.—H. R. Owens, chairman; J. C. Roberts, secretary; T. B. Dilts, G. W. Barager, E. W. Judd, E. E. Bach, J. L. Simons, Dr. M. J. Shields, H. P. Dowler, Dr. W. S. Rountree.

Committee 7—Hospitals and safety stations.—Dr. M. J. Shields, chairman; C. O. Roberts, secretary; T. I. Stephenson, Dr. J. W. Parshall, G. H. Hawes, R. B. Moss, John Meagher, Dr. W. S. Rountree, Dr. Ralph McHenry, G. W. Hutchinson, Dr. G. H. Halberstadt.

Committee 8—Permanent organization.—H. M. Wilson, chairman; H. A. Wadsworth, secretary; J. P. Reese, Dr. W. S. Rountree, Dr. A. F. Knoefel, Austin King.

RESOLUTIONS ADOPTED BY THE CONFERENCE.

The chairman of the conference was directed to refer the various resolutions recommended for adoption by the several committees to a committee on resolutions. The committee appointed comprised the following: Dr. F. L. McKee, chairman; J. W. Paul, secretary; B. J. Matteson, Dr. W. S. Rountree.

The resolutions, embodying minor amendments made as a result of general discussion, were reported to the conference on the morning of September 25, and were adopted. The phraseology of the resolutions as here printed has been slightly changed so that they appear as direct conclusions, as follows:

RESCUE APPARATUS AND RESCUE TRAINING.

1. Breathing apparatus used for mine-rescue and mine-recovery work should be of such types as have passed the tests of the Bureau of Mines.

2. The course in rescue training as outlined by the Bureau of Mines schedule for educational purposes and for familiarizing miners in the use of the breathing apparatus should be followed, as should the recommendations of the bureau pertaining thereto, to the extent that for actual mine-rescue work supplemental and practical training of two hours each should be taken at intervals of not more than three months.

3. All mine-rescue stations should be equipped with at least five breathing apparatus and the necessary accessories for the continuous operation of the apparatus for 24 hours, and at remote stations 48 hours. Such equipment should be so located as to admit of its assembly in one hour at a central point for emergency use.

4. The keeping of birds and mice at rescue stations for the purpose of detecting carbon monoxide is desirable.

5. All persons before being admitted for rescue training should present a medical certificate qualifying them for rescue work and stating that they are free from contagious diseases.

6. The Bureau of Mines should be requested to prepare lists of stations, together with the names and addresses of the persons who have completed training with mine-rescue apparatus, and to list the names on a roll of honor to be submitted for publication by the press in the various mining localities. Mine-rescue volunteers should be compensated for services and should be considered when promotions are being made.

7. The smoke room is best adapted for the first course of training. In addition to the smoke-room training as prescribed by the Bureau of Mines outline, supplementary practice to be taken underground is desirable.

The program subjects "Physiological Effect of Pressure on the Head Due to Wearing Helmet" and "Diet of Men Doing Rescue Work" having been referred to the general conference, were by it referred to a special committee of surgeons and rescue men, to be appointed by the chairman and to make report to the next annual meeting.

RESCUE OPERATIONS.

8. In mine-rescue work untrained men should not be permitted to use breathing apparatus except when such course is necessary in the saving of life and will not unduly jeopardize the life of the rescuer. In selecting untrained men, those should be chosen who are readily amenable to discipline.

9. A desirable outline of procedure before and after entering a mine following explosions or mine fires is as follows:

OUTSIDE ORGANIZATION.

a. All openings should be carefully guarded.

b. There should be a man in charge of outside arrangements who should see that ventilating appliances are put in readiness for operation when required.

c. Competent men who can be relied upon to obey the orders given them should be stationed at all mine openings.

d. A competent person should be placed near the mine entrance to examine all safety lamps before they are allowed to be taken into the mine.

e. Some specified person should be placed at the mine entrance to record all persons going into and coming out of the mine.

f. Proper food and shelter should be provided for parties engaged in rescue work.

g. A physician should be on hand while rescue parties are in the mines.

h. Around all openings there should be established safety lines inside of which no open lights should be allowed.

i. A man should be placed in charge of the rescue squads to organize and have them ready to enter the mine when called upon.

INSIDE ORGANIZATION.

a. A man should be placed in full charge of the inside operations on each shift.

b. An advance squad under a competent leader should explore the workings in advance of the stretcher squads and the other squads who are advancing the ventilation, making repairs, etc. The squads should advance in the following order: (*a*) Breathing-apparatus or advance squad; (*b*) stretcher squads; (*c*) temporary-ventilation squad; (*d*) material squad; (*e*) permanent-ventilation squad.

c. An inside station should be established as a base of operations. A competent person should be placed in charge to reexamine all lights before they pass to the interior of the mine.

d. A telephone communicating with the surface should be established at the inside station and should be carried forward as fast as possible.

e. A doctor provided with necessary supplies should be present at the inside station.

f. No person should go in advance of the ventilating current except the advance squad, which should examine the atmosphere for gas, and examine the return air current frequently for indications of fire; also for any other dangers which are likely to exist.

g. The advance squad, when proceeding into the mine, should mark all unexplored openings with a danger sign.

h. Strict discipline should be maintained at all times.

10. The maximum distance rescue crews may safely proceed beyond fresh air should, owing to the different conditions in different mines and the hazardous work undertaken, be left to the decision of the trained official in immediate charge, in conjunction with the mine officials, who should be governed by the probability of being able to save human life, using the time limit on all explorations.

11. Except when absolutely necessary, the shift of rescue men should not exceed two hours. This should be followed by not less than six hours' rest.

The program subjects, "Cumulative Effect of Imbibing Poisonous Gases," and "Use of Stimulants," were referred by the general conference to the committee on first-aid training for future report.

SAFETY LAMPS AND ELECTRIC LAMPS.

12. No open light should be used.

13. In coal mines only such types of safety lamps and electric lamps as have passed the tests of the United States Bureau of Mines should be used.

14. Electric lamps unaccompanied by safety lamps should not be used unless the party is equipped with breathing apparatus.

15. Both safety lamps and electric lamps should always be used up to the point where breathing apparatus is put on, and beyond such point no safety lamps should be used except one carried by the leader of the party; if explosive gas is present in dangerous quantities, even the leader should not carry a safety lamp.

FIRST-AID METHODS.

16. The Sylvester method of artificial respiration is the preferable method, provided no injury of the person to be resuscitated prohibits the use of this method.*

17. In the case of dislocation of a hip or a shoulder, the dislocation should not be reduced, but the limb should be fixed in the line of deformity.

18. A man injured with a broken back should be handled as little as possible. If found in any other than a recumbent position, he should be kept in that position; if found in a recumbent position, posterior splints extending from the head to the feet should be applied, or he should be laid on rolled blankets.

19. In the treatment of any fracture of a long bone it is necessary to apply splints long enough to fix the joint above and below the fracture; for example, if there is a fracture of the lower part of the leg, the splints should be applied so that they extend below the ankle and above the knee. Long splints are especially to be recommended in fractures of the legs. The forearm and arm splints designated by Dr. M. J. Shields and described in American National Red Cross textbook on first-aid, page 71, are commendable.

20. It should not be the duty of a first-aid man to reduce a dislocation except of the lower jaw or a finger.

21. The triangular bandage should be used in preference to the roller bandage.

22. A first-aid man should not wash a wound. The application to a wound of any foreign substance other than a sterile substance should be condemned.

23. In the case of a severe hemorrhage that is being controlled by a tourniquet, if no sterile or antiseptic dressing is available, no dressing should be applied to the wound.

24. An injured person should be carried feet first on a stretcher, unless there be some reason for a contrary method.

25. In the case of an electric shock the current should first be either cut off or short-circuited, if possible; if not possible, the rescuer should insulate himself and remove the patient from the body that carries the current. The treatment of electric shock prescribed in Miner's Circular 5 of the Bureau of Mines should be adopted.

26. In moving an injured man his injured side should be next to the first-aid men lifting him. A rescue corps bearing an injured man should in all cases have the right of way to the surface.

Topic A 12, "Development and Acceptance of New Methods and Dressings," was, by resolution 27, referred by the conference (see p. 39) to the executive committee.

FIRST-AID TRAINING.

28. The duties of a first-aid man are to act intelligently and efficiently between the time of an accident and the time the injured man is placed in the hands of the physician or surgeon or in a hospital.

29. The program subjects "Instruction; Its Kind and Amount," "Training and Practice; Kind and Amount," and "Systems of Training: Red Cross, Navy, or New Combination; Value of a Standard System for Miners," should be treated as one subject, and the Red Cross system of first-aid training should be adopted as the standard.

* The author suggests that judgment regarding the preference given the Sylvester over the Shafer method be suspended pending report on an exhaustive investigation of the subject for the Bureau of Mines by a committee designated by the American Medical Association.

30. Successful first-aid work at any given mine must have the personal interest of the operating officials, the financial support of the mining company, and the cooperation of the mine physician, surgeons, and employees.

31. Every mine should at all times have a sufficient number of first-aid men on duty to take care of any persons injured during any of the 24 hours of the day.

FIRST-AID CONTESTS.

32. For the larger companies having a great many mines, intracompany contests are to be preferred as against exhibitions, whereas for the smaller companies, operating only one or two mines, intercompany contests are preferable.

33. The method of judging should be by a system of discounts, the following discounts to prevail at all contests:

Discounts for judging first-aid contests.

	Points discounted.
1. For not doing most important thing first	2
2. For captain's failure to command men properly	2
3. For slowness in work	2
4. For failure to entirely cover a wound	2
5. For wrong artificial respiration	2
6. For loose splint	2
7. For not padding splints properly	2
8. For loose bandage	2
9. For bandage too tight	2
10. For "granny knot"	2
11. For awkward handling of patient on stretcher	2
12. For lack of neatness	2
13. For assistance lent by patient	3
14. For tourniquet too tight	3
15. For failure to stop bleeding	5
16. For not treating shock	5
17. For failure to be aseptic	10

34. Each contest should consist of not more than five events. The judges, who should be first-aid men and surgeons, should be of sufficient number, so that one judge does not inspect in any single event more than three teams, and preferably less. These judges should inspect sets of teams progressively.

35. In all contests speed should not be an essential element, but a certain time should be allotted to each event. Failure to finish in the allotted time should be discounted 1 point for each minute over time.

HOSPITALS AND FIRST-AID STATIONS.

36. Underground surgical hospitals are unnecessary, but there should be deposited at different points in the mine a sufficient number of first-aid packages properly equipped with first-aid emergency dressing. In addition there should be located central first-aid dressing stations at the bottom and at the surface opening of the mine.

PERMANENT ORGANIZATION.

The last resolution of the conference is presented below in substantially the form in which it was unanimously passed.

37. Resolved that this first national mine-rescue and first-aid conference should be made a permanent organization; that to this end there should be

created a temporary committee to draw up a constitution and by-laws and otherwise complete the organization; that the committee should be composed of 10 members, the presiding chairman of this conference, H. M. Wilson, to be the president and C. S. Stevenson the secretary, the vice president to be elected and the other seven members of the committee to be appointed by the president.

A draft of a constitution for the proposed permanent organization is presented in a subsequent section of this report.

OPENING SESSION OF CONFERENCE.

The opening meeting of the conference was called to order by H. M. Wilson at 9.20 a. m., September 23, 1912. He announced that Dr. J. A. Holmes, Director of the Bureau of Mines, regretted that urgent official engagements elsewhere prevented his being present to welcome the delegates. By request of the director, Mr. Wilson welcomed the delegates, his remarks being as follows:

ADDRESS OF WELCOME.

The Bureau of Mines is immensely gratified at the splendid response made to its invitation. It is not only gratifying to the bureau, but it must be equally gratifying to yourselves and to those you represent that so large and so representative a body of men have gathered here this morning. We have here delegates representing districts scattered from eastern Pennsylvania to Montana and from Michigan to Alabama, and we believe that such a representation makes this meeting an assured success. We want to make this a real heart-to-heart business talk on the details of those subjects outlined in the program before you.

Having welcomed you on behalf of the Director of the Bureau of Mines, I now, at his request, assume the duties of and will speak as temporary chairman.

CHAIRMAN'S PRELIMINARY REMARKS.

The chairman continued his remarks, as follows:

I am sure you will all agree that though we want this conference to be conducted by you, the representatives of the mining industry concerned in first-aid and mine-rescue work, it was necessary that some one take the initiative in organizing for business. Therefore I know you will appreciate the motive of the Bureau of Mines in asking certain of its officials to act temporarily as chairman and secretary and in submitting a tentative program. Subsequently I shall turn the proceedings over to you for such action as may suit your pleasure.

First, however, I believe that there should be a little preliminary discussion in order that we may settle our minds upon the subject in hand after the fatigue of the long journeys which many of you have undergone. I will accordingly outline the way the subject impresses me and will then call upon spokesmen of the different interests here represented to express briefly their opinions as to what this conference may do.

By invitation of Vice President Richards, of the Philadelphia & Reading Coal and Iron Co., I was present last Saturday at East Mahanoy Junction, Pa., at the company's annual field meet, certain features of which impressed me greatly.

There were present nearly 2,000 employees of the company, representing all classes of employment; and as I glanced over them I recognized many of the higher officials of other great mining companies, and I wondered as to the amount of the pay roll represented by the employees there idle in the cause of humanity. The Philadelphia & Reading Coal and Iron Co. had on that day, in the interest of this great movement, withdrawn from work many of the best men in its 66 collieries, had prepared a dinner—not a luncheon—for 2,000 people, had employed several bands of music, and had run a half dozen special trains. The pay roll alone represented by the idle men could not have been less than $20,000. The interruption to the business of a company that was thousands of tons behind in its contracts, with other items of expense involved, must have made a total loss considerably in excess of $50,000. This expenditure measures the interest of that one great company in the increased safety of its employees. On the other hand, the long weeks of training by the miners and their earnest work for the success of the field events clearly showed their appreciation of the company's efforts and the importance of the first aid to themselves and to their fellow miners. And the same sort of thing is now going on all over this country.

We who have had experience in this work find that, aside from the expense involved, it is quite an effort to maintain the interest of the men, and various expedients must be devised to keep up an interest that will otherwise soon die out. In the competitive meets appropriate and valuable prizes are presented to the winners, and some of the mining companies have rewarded their winning teams by sending them to this very meeting; some were sent to the first national mine-safety demonstration last fall. One of the most important things to be considered here is what can be done to retain the interest of the men and to induce them to develop the elementary stage of the training. Another is the adoption of some standards in first-aid and mine-rescue training methods. The program outlines many of those subjects regarding which there are differences of opinion, and we hope that in the deliberations of this conference many of these differences may be swept aside and replaced by a clear understanding of each other's methods. There are several systems of judging first-aid contests, and it is utterly impossible for the men of one district to compare the merits of their work with that of another district by published ratings. We should this week get together and establish a standard form of judging.

It is undoubtedly clear to all of us that standards of training and judging are desirable, for they will give us standards of comparison that will lead to the establishment of the work on a systematic basis. We have all been working independently, and we have not been able to profit by the experience of others, but we believe that through the medium of this conference it will be possible for us to arrive at a common understanding. The Bureau of Mines, if so requested by this conference, will itself undertake the publication of your conclusions on these several matters.

I now present to you a man who has had a wide experience in this work. Others of you may have had as great or greater experience locally, but none, I believe, have had the opportunity of studying in so many different parts of this and foreign countries the subjects that we shall consider. I introduce J. W. Paul, who is in charge of the mine-rescue and first-aid operations of the Bureau of Mines.

ADDRESS BY J. W. PAUL.

Mr. Paul addressed the conference as follows:

Mr. CHAIRMAN AND GENTLEMEN: The testing of breathing apparatus by the employees of the Bureau of Mines and the outline of mine-rescue and first-aid

training therewith used by them have been patterned largely after the practices abroad. We have endeavored to profit by the work done in European countries by sending representatives of the bureau to visit their stations and to ascertain the details of the manner in which they test the apparatus and the details of the methods followed in training men in the use of the apparatus. We have been training men now for about four years, and a large number of miners have been trained throughout different part of the United States. As we have received valuable suggestions from those interested in the work we have freely adopted them. These suggestions do not always come as frequently as we desire, and we recognize that the training we give is not as extensive as it should be, as compared with the course of training at some foreign stations where regular rescue crews are maintained, and it is hoped that this conference will enable us to adopt a satisfactory standard of training for the miners. This will serve as a basis of training the miners in the principles and proper use and care of the apparatus.

There are some differences of opinion as to the amount of training that a man should have before he should be certified as qualified to participate in actual rescue operations. We should like your suggestions as to the number of hours a man should be trained with oxygen, and as to whether training without oxygen is valuable. Then, too, the physical standard of the man who is to take the training is as yet not a definite matter. These are some of the points that we hope the deliberations of this conference will settle with some degree of satisfaction.

The operations of the Bureau of Mines in this work have covered practically all of the mining fields of the United States, and have led to the adoption of various types of rescue apparatus by many mining companies and the establishment of many rescue stations by different companies and States. We have schedules outlining the training of miners in rescue and first-aid work, a form used for the physician's examination of the men, and forms for recording the training work. They will be at your disposal for consideration, and we hope to have your criticisms and suggestions on them. Mr. Wilson has largely covered the field of first-aid work and pointed out the need of unification in that work. We have profited greatly in our work of first aid to the injured by the experience of those interested in this work in the anthracite field, in the Birmingham (Ala.) field, and in the Colorado field, and by the work of the Red Cross.

During the last year the bureau had at its several rescue cars and stations 53,631 visitors and 36,000 auditors at lectures. Of these 4,700 wore the apparatus, 2,332 were given first-aid training, and 2,000 were given rescue training. These figures will give you some idea of the extent to which the Government training has extended.

GENERAL DISCUSSION.

At the conclusion of the foregoing remarks by the chairman and Mr. Paul on behalf of the Bureau of Mines, the chairman threw the meeting open to more general discussion, introducing first Mr. J. P. Reese, with remarks as follows:

The CHAIRMAN. Having heard from a man who speaks for training in general throughout the country, I now take pleasure in calling upon one who represents the operators' viewpoint—a man who has had an unusual experience in that he has been a mine worker; he has also been connected with State commission work, and he is now the superintendent of the Superior Coal Co., of Gillespie, Ill. I introduce to you John P. Reese.

J. P. REESE. Gentlemen, I came here to learn and not to instruct. This is my first meeting of this nature, and I hope to be a better authority on mine-rescue and first-aid work on next Thursday, which marks the conclusion of these proceedings. I realize, however, that we are all new in this business and that we should not hesitate on account of our lack of experience to express our views. Our company, as do most of the companies in this country, realizes the importance of having trained men at hand to take care of the unforeseen emergency. We as a company have not progressed far in this work, but we have made a beginning. We have sent a few dozen of our men to be trained at the Springfield station, where they have obtained their certificates, and these men are meeting weekly and training other men in turn. We have equipped our two operations, one in Illinois and one in Iowa, with the rescue and first-aid materials that we have been advised are necessary for the successful operation of this work. We have at each place helmets, pulmotors, electric lights, safety lamps, and first-aid materials. We have no statistics on the practical results as yet, but we do know that in our mines the nearest first-aid man is always summoned immediately after the injury of a man. We believe that the suffering is thereby reduced to a minimum during the wait for the arrival of the physician, a period that at best is from one-half an hour to two hours. We believe that all of the money we have thus far spent has been amply repaid by the results we have obtained, and we hope for still better results in the future. We believe that in time it will be demonstrated that even from a financial standpoint this effort has been a good investment; we already know from a humanitarian standpoint that it more than pays. We need the first-aid work all of the time.

I hope to receive enlightment upon the best expedients to be used in retaining the interest of the men already trained and in extending the work to as large a number as possible of their fellow workmen. We have no standards to which we aspire; every one has had to work out his own plan. But we have succeeded in getting a great number of our best men interested. The miners' union should take more interest in this work, from their national president down to the local pit committee, and if this meeting can present this matter forcibly to the dominant powers of the miners' organization it will do a great favor to the mining industry.

The movement we represent here to-day is one that should receive the cooperation of all mining interests. When the miners fully realize that this work is being done from a humanitarian standpoint alone, we will have no trouble in training great numbers of them and in keeping them thoroughly interested in the work. So far I can not say that any real assistance to the movement has been given by the miners themselves. In reality the pioneers of the movement have been scoffed at. But we have had the rescue apparatus and have used it under actual rescue conditions. Two weeks ago three members of our organization, together with myself, explored a large area that we had holed into and which was filled with gas in which men could not long live. We explored six entries, keeping three men at the fresh air for relay and any necessary rescue work. We have used the apparatus in two other squeeze territories that were full of gas—one of them having been shut off for over two years—and we had no difficulty at all; most of the boys that used the apparatus had been to Springfield and taken the instruction. I personally had had no training except that which we took at home; but you will find that the breathing apparatus, if used with a little common sense, is a reasonably safe instrument; and although we were not compelled to build any fire walls nor carry out unconscious men, we found that we could do such things with the apparatus if we had to. We also explored one territory that had had the ventilation off of it for over a year and

was filled with black damp. We remained in the territory for 30 to 45 minutes, found out all we wanted to know, and came out with no bad effects.

It is my opinion that the coal companies will all cooperate in this first-aid and rescue work as soon as it is forcibly brought to their attention. It will be hard to interest the employees and hard to keep them interested after once attracting their attention, and undoubtedly we will have a few more actual examples of avoidable fatalities before the benefits of the system are accepted by all. However, I look forward to the day, in the not distant future, when every coal mine and all other mines will have a large percentage of their employees trained to administer first aid. Then, if we are unfortunate enough to have explosions and mine fires that call for rescue work, we will be able to reach the danger quickly and not have to wait for one of the rescue cars to come from an adjacent State or for the crew to come from the rescue station. We will have a trained corps of men and a reasonable amount of equipment on hand, and we will be able to do the work with the help of our neighbors, and when the big fellows get there with the cars they will pat us on the back and say, "You have done all that we could have done." I hope that within the next few days we who have assembled here will become well acquainted and shall leave greatly profited by each other's association.

J. C. ROBERTS. Do you have to pay these men to take the training?

J. P. REESE. That's a family secret. I'll say that we have paid in part, not in full. We pay them for actual first-aid work. We do not pay for their evening meetings, but we did pay all of their expenses when they were taking the training at the Springfield station.

The CHAIRMAN. The question that Mr. Roberts raises and that Mr. Reese has just answered is, I think, a very important one, and I hope that the committee concerned will deal with it very fully, as it has never been definitely settled as to whether it is justifiable to pay the men for this training. I also noted with pleasure Mr. Reese's statement of his ambition to be independent of State or Government assistance in the time of disaster.

I now take pleasure in introducing to you a physician and surgeon who has had much to do with the development of the magnificent first-aid work that many of us have seen in the Birmingham, Ala., district. He represents the Tennessee Coal, Iron & Railroad Co., and it is through him that the first-aid work has kept apace with the other great developments of that company. I take pleasure in presenting Dr. W. S. Rountree.

Dr. W. S. ROUNTREE. We are here to consider preventable disasters and to work out the methods for caring for the injured in the cold light of truth. Let us not deceive ourselves in the least; let us apply the rod of self-punishment without flinching and try with might and main to correct, as far as possible, our corrigible faults. I speak of the elimination of the preventable disasters as if we had arrived at that height of perfection at which there are no longer any disasters, or only few, to eliminate.

From the standpoint of the doctors and surgeons, let me say that he is unfortunate indeed who finds himself in one of those old-school mining camps that do not have first-aid men to support him in time of disasters. Moreover, this does not apply only to mines, but to communities where steel mills, furnaces, railroad shops, and quarries are located or to any place where a great number of men are collected. It has been my good fortune in my small individual way to organize and train first-aid and mine-rescue corps in Alabama and we feel that our men are as efficient in this great work as the more seasoned teams of the anthracite regions.

I have not been engaged in the work very many years, but what we have done in Alabama seems to be bearing good fruit. I am eager to continue this work,

and I know by coming to this meeting I will be furnished with food for thought. I can remember that it was only a few years ago when it was a common occurrence for men to lose their lives in mills and mines of our State owing to the absence of first-aid men in the vicinity of the accident.

Gentlemen, the conservation of human life and human energy should be our watchword. It is by far the most important of all questions before the American people to-day, and I believe that the first-aid movement is a step in the right direction. We physicians and surgeons must push this work along. It devolves upon us to do the greater part of the work by teaching the layman how to prepare himself for the saving of the life of his fellow workman, and thus we will render an everlasting service to suffering humanity.

In teaching the layman first aid to the injured you will find that he will grow very tired, because it is a dry subject and especially so to the miner, but by persistent efforts and by injecting social features into the training you will keep up interest. By and by the beauty of the work will begin to unfold itself and all will experience genuine pleasure through their ability to relieve suffering.

I might state in passing that the knowledge I try to impart to the layman by lectures and demonstrations is based upon the teaching of modern surgical practice and common sense. The suggestions are not elaborate or extensive, the ruling consideration being simplicity. Much that is ordinarily embraced in the teaching of surgery is omitted in an effort to select that which is essential and most helpful. Technical terms are avoided and slight attempt is made to teach anatomy, physiology, and hygiene. We believe that in a few years first aid will be taught in our public schools and in all places where great bodies of men are assembled in industrial pursuits.

In order that first aid may be successfully taught you must have the personal interest and financial support of the company officials and the hearty cooperation of the physician, surgeon, and employees. In my State some of the companies and the doctors are slow to take up this much-needed work. I must say, however, that in my company the interest is universal—from the lowest official to the highest—and by virtue of that fact first-aid and mine-rescue work is a success with us from every standpoint.

Our company a few months ago gave quite a few of our men substantial checks for some of the work they had done. It happened that two of the men in one of our mines were shocked by electricity, and two of our first-aid men reached them in time to resuscitate them. They were one hour in resuscitating one of the unfortunate men, and an hour and a quarter elapsed before the second was brought to consciousness, but the two are to-day living and working. This is only one example of what men trained in this work are able to do.

I draw no color lines in teaching this work. We have black men as efficient in giving first aid to the injured as are the white men. Again, the beauty of this work is not only taught to the men but to the women and the girls and to the boys who are soon to take our places in the great industrial pursuits of this age. Many women come to me and tell me of cases in which they have been the first who have been asked to take care of an injury, and they have since asked me what they should have done. It is within the province of womankind to relieve suffering, and in the future we are going to teach them the work of first aid to the injured in our mining camps. A few months ago an accident occurred in which 15 men were killed and many injured, and a first-aid woman undertook the work of caring for the injured, and I am pleased to say that her services were the most beautiful of any so engaged after the accident. At the recent demonstration of the International Red Cross at

Washington it was my pleasure to witness the work of four teams of women so trained, and their work was very creditable indeed.

Gentlemen, think what first aid means to the wives and children, as well as the men themselves; how it may save many a man from being crippled for life; more than this, how it may mean the difference between life and death itself. The time has come when we must be teachers as well as doctors and mine experts—forgetting selfish motives, remembering the great fundamental principles embodied in "Do unto the other man as you would that he should do unto you."

One of the things that I hope this conference will do is to adopt certain standards of first-aid training, the lack of which has heretofore caused friction in our first-aid meets and competitions.

The CHAIRMAN. I have been very much interested in this address by Dr. Rountree. It has brought to my mind for the first time the fact that it is entirely possible and practical for the women of our mining camps to be of service in this great effort for reducing the loss of life in our mines. I see no reason why the work can not be extended to women with the full expectation that we shall be greatly repaid by our efforts. It is certainly within the province of womankind to take care of the injured, and as an example we have only to recall the great work done by the Red Cross nurses on the field of battle. I hope that the proper committees will give this matter of the training of the women of our mining camps the consideration that its importance seems to me to demand.

I now take pleasure in introducing to you a man who represents the inspector's point of view in this work. He is one well known in the inspection force of the bituminous district of the State of Pennsylvania. I take pleasure in presenting F. W. Cunningham.

F. W. CUNNINGHAM. Mr. Chairman and gentlemen, I did not come to this conference with the expectation of addressing you, but I came in the hope of listening to the wise counsel of others and profiting thereby. I fully appreciate that we are in need of some standard system for this work. I find different methods at each camp where first aid has been inaugurated, and it may never be possible to have a uniform quality of work throughout the whole country. Men will insist on performing their work by that method which they know how best to apply.

I believe there is great good done by first-aid exhibitions. They not only maintain the interest of the regular rescue crews, but they are of positive instructional value to the spectators. For instance, the other day in one of our mines a boy was injured while trying to jump aboard a trip of moving cars. The trip rider was a Hungarian who had seen the first-aid teams in operation, but who had never been given any actual training himself. He, however, administered first aid from his memory of the work that he had seen the first-aid crews do; he claimed that the boy was unconscious for half an hour and that blood flowed from his nose and mouth. Just what process he used to restore the boy he could not explain, but I do know that he must have made a heroic effort and that the boy is living and well to-day.

I believe it is important for this conference to adopt standard methods of judging the work in these first-aid meets. We have no assurance that equal rating as published in reports from Alabama, Colorado, and Pennsylvania means that the work of the teams in those States is of equal value.

The Pennsylvania law requires that all necessary first-aid materials be kept at each mine. We have, however, gone a step further than they have in some States, in that we have decided that since we must have the first-aid materials at hand we should also have at hand men who know how to apply such mate-

rials. The only compulsory first-aid work in this State is that which requires men working around electricity to be able to apply resuscitation in the event of shock. I find that in the bituminous district of Pennsylvania the higher officials of the companies stand nobly by this work; our greatest trouble is that the officials at the mines do not give the proper cooperation. I believe that the bituminous field of Pennsylvania deserves credit for the work it has already done. Still greater advance is possible, but in many localities of this field you will find a good proportion of the men trained and capable of taking care of an injury to their fellow employees. The mining department of the State of Pennsylvania believes that fatal accidents due to falls of roof and other similar causes may be cut in two by the work of properly trained first-aid men, and it is unnecessary for me to say that the cooperation of that department is assured in anything that this conference may do that will assist this great cause.

The CHAIRMAN. Perhaps no other agency has done more to develop first-aid work than has the American National Red Cross, and to one man more than to any other is due the credit for the great work that the Red Cross has done in this line. He it was who trained the first crew instructed in an American mine; he is the man who trained the first-aid men of the Bureau of Mines; and it is largely through his efforts that we have seen this work developed in the United States to its present standard of excellence. I take great pleasure in introducing to you Dr. M. J. Shields, of the American National Red Cross.

Dr. M. J. SHIELDS. Mr. Chairman and gentlemen, an invitation was presented to Maj. Charles Lynch of my department to be present at this conference, but due to the pressure of other business he found it impossible to be here, and I have been ordered to report in his absence. I am very glad of this opportunity to be present. I see here many of the friends who have cooperated with me in this work during the past three years, and the occasion is indeed a pleasant one.

As Mr. Wilson has pointed out, I believe that the object of this conference should be the practical standardization of first-aid methods, and, if these contests are to be continued, the standardization of the methods by which they are to be judged. For instance, I have always been an advocate of the triangular bandage. The roller bandage requires a great deal of skill for its proper application; indeed a great many physicians in the United States to-day can not correctly apply the roller bandage—that work is usually done by the nurse; yet we have advocates of the use of the roller bandage in first-aid work. I have in mind a case where an injured man was delayed an hour and a half in his arrival at the hospital because the first-aid man wanted to apply a fancy roller-bandage dressing. This could have been done with a triangular bandage in two minutes; and because of the facility with which the triangular dressing is applied I believe we should all advocate its use. First aid, as I understand it, is the connecting link between the time a man is injured and the time he arrives at a hospital. I feel fully convinced that it is a mistake to teach the rank and file the use of the roller bandage.

One of the important things for this conference to do is to standardize the methods of judging first-aid contests. In the past it has been a common thing to have friction at these meets because of the difference in the teaching of the various teams. If we have suitable standards adopted here men will in the future know what to expect in these contests and friction will be avoided. I shall be very glad to assist in this work in any way that I find possible, and I believe the doctors all over the country will welcome any information as to how they should teach and what they should teach.

In the line of compensation for first-aid men I believe that the system used by the company where I organized this work in the anthracite field was one deserving attention. Men who were faithful attendants and skillful artisans in this work were given precedence in the matter of promotion in the mine, but no direct compensation was made for the efforts that the men directed to the work. I have often made the statement both to miners and railroad workers that it is they who derive the greatest benefit from this work. It is their lives that are saved by the efforts of the first-aid men. I tell the men that it is the best insurance which they can have if they are engaged in a hazardous occupation; I tell them that with the proper administration of first aid they will get well from a serious injury 10 to 60 days earlier than without it.

The matter of judging has given me greater worry than any other feature of this work. I remember that in the first contest in the anthracite field in 1906 we gave 33⅓ per cent for correct method, efficiency 33⅓, and time 33⅓ per cent. By giving so large a percentage to the element of time we found that all men were inclined to speed up their work and thereby put on an inefficient dressing. Gradually this element of time was cut down as much as possible, so that in our meets at the present time men proceed with all the care that the seriousness of the injury demands. T. B. Dilts and a local physician at Greensburg inaugurated at a recent contest a system that will eventually overcome much of the wrangling often seen at these contests. Mr. Dilts's plan is to have five physicians do the marking, and after each dressing these physicians change to a different team, and in this way each team gets the benefit of the marking of different physicians. It will be impossible, perhaps, to create a table of discounts that will fit all circumstances. H. M. Wolffin, one of the engineers of the Bureau of Mines, has suggestions in this connection that I believe are excellent.

In the matter of maintaining the interest of the men, after extended experience throughout the United States, I find that, after all, it is simply a question of what interest the officials may have in the work. Where the officials are passive the men are passive, and where the officials are active the men reflect their interest.

It has been my experience that we should not teach first-aid men to wash wounds. Prominent surgeons to-day do not wash wounds as they did formerly. I have told men that iodoform is the medical skunk, and that that popular disinfectant, peroxide, is nothing but wind and water; that the best they could do is to use nothing but the local air and keep impure materials and disinfectants away from the wound. I tell them that germs are, in a way, like flies; they are on your clothing and your hands, but they do not fly around in the air to any large extent.

The CHAIRMAN. I believe that Dr. Shields's remarks should be the keynote of this conference, and that the committee should give close attention to this matter of the triangular bandage versus the roller bandage, and to the proper protection of a wound to prevent infection. I was much impressed with the reason assigned for the use of the roller bandage in one field that I recently visited. It was that the men were taught the use of the roller bandage as a postgraduate course to the simpler and more elementary triangular bandage, and that it was done largely to retain the interest of the men in the work over long periods.

Dr. M. J. SHIELDS. The Red Cross is about to promulgate what we call a "medallion course" of instruction, which will be very nearly equivalent to the work of the St. John's ambulance corps in England. This medallion crew will have a personal connection, as it were, with the army in that these men will

be called upon in time of need, but of course may use their pleasure as to enlistment. This has been done simply to retain the interest of the men. The instructors of others in first aid should undoubtedly have had an advanced course that is the equivalent of our medallion course; they will then be enabled to teach others from a broader point of view.

The CHAIRMAN. The chair now asks the pleasure of the conference, and will entertain any motion regarding procedure.

T. B. DILTS. Gentlemen, I move you that the conference be divided into committees, as outlined in the program, and that these committees be appointed by the chair.

Dr. A. F. KNOEFEL. I have heard nothing yet as to the organization of this conference, and I think that it is well that we should consider the appointment of a committee by the chair for organizing this body permanently. We need an executive committee, to which all matters that may come in dispute from time to time may be referred.

T. B. DILTS. I amend my motion to include the appointment of eight committees by the chair, to take care of those subjects outlined in the program, and a committee on organization, as Dr. Knoefel has suggested.

The amended motion was unanimously adopted by the conference.

J. P. REESE. I move that the organization committee be composed of five individuals, and that all other committees be composed of three individuals.

This motion was not seconded, but was followed by discussion as to whether the first meetings should be in sections or whether the individual committees should meet at once. It was finally decided that medical and first-aid men should meet in joint conference, regardless of committee assignments, during the afternoon of September 23, and that mining men and those interested in rescue work, regardless of committee assignments, should also meet separately on the same afternoon; and that in the evening and during September 24, mining men and surgeons who were members of committees 5, 6, and 7 should meet in joint conference.

J. P. REESE. I move that all committees be composed of a minimum number of three, except the organization committee, which shall be composed of a minimum number of five.

Unanimously carried.

Dr. M. J. Shields moved that each committee should have a chairman and a secretary appointed by the chair, and Dr. A. F. Knoefel amended the motion to read that the chairman be elected by each committee, and that the secretary be appointed by the chair from a member of the Bureau of Mines staff. The motion as amended was unanimously carried.

J. P. Reese moved adjournment for 30 minutes, in order that the chairman might make selections for the various committees.

The chairman called the meeting to order at 12.20 p. m., and announced committee appointments as mentioned in a foregoing section.

PROCEEDINGS OF JOINT MEETING OF COMMITTEES 1, 2, AND 3.

Committees 1, 2, and 3 met jointly at 2 p. m., September 23. J. W. Paul was elected chairman and D. J. Price was elected secretary.

The CHAIRMAN. The purpose of this joint committee meeting is to discuss breathing apparatus, training courses, and the operation of crews in mine-rescue and recovery work.

There is placed before you for your perusal and criticism the bureau's publication on rescue apparatus,[a] and an outline of the bureau's course of training with breathing apparatus. We earnestly solicit your criticism and ask for any suggestions you may care to make that will aid us in making our work more efficient.

The meeting is in your hands and the chairman will be pleased to entertain any motions, resolutions, or discussions you may present.

On motion by Mr. Reese, the secretary, read extracts from the Bureau of Mines miners' circular above mentioned relative to the use and care of mine-rescue breathing apparatus.

J. C. ROBERTS. I think it bad to instruct rescue men to remove bottle or cartridge from their own apparatus.

CHARLES ENZIAN. I helped originally to suggest the idea of training men to remove the cartridge.

AUSTIN KING. A trained man is one who knows how to act, one who will never enter a mine to do rescue work alone. I think it is a very good idea to train the men to replace their own oxygen bottle, so that they can act together in case of an emergency.

J. P. REESE. I am opposed to the doctrine that two men could not go in alone. It is not necessary for four or five men to enter at one time. I have had experience with a party of three men which worked out successfully.

HENRY OWENS. A crew of not less than four men is safe, and in preparation of a man for emergency work I have trained the men to remove the bottle themselves.

J. L. SIMONS. I believe it is necessary to have at least four men in a crew.

WILLIAM CONIBEAR. Referring to removing oxygen cylinder and cartridge, I see no advantage in a man doing it himself. I think it is wise to have no less than five men in a rescue party, in which case a man could be assisted in removing the oxygen cylinder and cartridge.

H. P. DOWLER. I think the question of how many men should constitute a party should be left to the operators.

H. R. OWENS. I believe you should always get five or six men if you can; not less than four.

G. H. HAWES. The Oliver Mining Co. has three stations a half hour away, with three men at each station, so that they can all be assembled in a short time.

J. L. SIMONS. I do not think two men can take out one man.

G. H. HAWES. Our practice work included the carrying out of one man by two men to the bottom of the shaft and to the cage. The man was laid on a car and pushed out—actually carried and dragged, tied to a rail—a distance of 300 feet from the bottom of the shaft.

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[a] Paul, J. W., The use and care of mine-rescue breathing apparatus. Miners' Circular 4, Bureau of Mines, 1911, 24 pp., 5 figs.

AUSTIN KING. The Frick Co. have gone carefully over this question, and I think has spent more money than any other company in the training-to-rescue business. We believe it is best to have 5 persons to a team, 1 of whom should be the captain. We have 36 rescue teams and 3 stations, and we believe that a team of 5 men is an ideal team. We try to make the training as difficult as possible so that the men will be thoroughly trained.

WILLIAM NISBET. I have had an experience in a mine and am not in favor of one or two men going into a mine. There had been a very large mine fire there and in order to recover part of the mine it was necessary to go about 700 feet and build two stoppings. While working one of the men was partly overcome and we had great difficulty in getting him out. There should be not less than seven men to go into a mine in a fire or an accident. I think we ought not to take a chance with one or two men.

AUSTIN KING. The truth of the matter is that we take only a few men from the outside, and they are men of exceptional make-up. Some outside men are afraid to enter the mine, and we try to pick out those that will, and have them on our rescue team. We try to get men who are not afraid to work in the mine when a disaster occurs. I suppose that we could get our 36 corps to one place in 3 hours if it was necessary.

B. J. MATTESON. In the West, in regard to helmet work, we have five stations and a car, which, in addition to four helmets and a pulmoter each, have a telephone with each crew. The telephone is taken in 4,000 feet, which is considered far enough for a helmet crew to go. This portable telephone is certainly a great help to the helmet men. It is similar to what linemen use and is a pocket telephone. It has one mile of twin-conductor cable, and the wire is wound on a reel with a handle, which makes it very easy to carry.

The earpiece is strapped to the carrier's head. It is a very satisfactory arrangement and very easy to carry.

AUSTIN KING. I agree with the gentleman that these telephones are a great convenience. In Pennsylvania they are sometimes of a portable character, but generally they are of the regulation mine-telephone type. There is no doubt but that they are a very great help.

WILLIAM LYNCH. I think a telephone would be a great help to the Bureau of Mines. Why not put the telephone lines in through their life line, and when we have a man in the mines we can talk to him if we desire? That is something new and is undeveloped as yet.

J. W. PAUL. The bureau has such telephone equipment on each of its safety cars. The helmet has a transmitter inside and the receiver fits over the ear in a manner similar to those used by telephone girls.

By the end of the week I hope to have on exhibition here a new type of telephone made by the Western Electric Co. This is a telephone by which one can talk by using the ordinary receiver, and by the transmitter, a little disk that is strapped on the neck, one can talk and be distinctly understood at the other end of the line.

In regard to the course of training calling for the removal and replacing of the regenerator and oxygen bottles of the Draeger apparatus, it is not recommended that men do this in actual mine-rescue work except in case of extreme necessity—in case a man finds that he can not get out of the mine without a renewal of his oxygen supply. If he finds that he is not able to get back and has no assistance, he might possibly be able to save his life if he could put a freshly charged bottle on his apparatus.

In regard to the number of men that should compose a rescue party, the Bureau of Mines has been insisting upon not less than five men to compose a

rescue crew, for the reason that if one meets with an accident, there would be four men to bring him out.

We think a man has a right to refuse to go into a mine alone, as he has no assurance that he will be able to get out alive; nor should only two men enter a mine filled with poisonous gases. There should be a sufficient number of men to guarantee confidence on the part of all the men.

Some of our men have had considerable experience in mine-recovery work, and when they have been on prolonged expeditions extending over four or five days or probably a week, making long journeys several times daily, they have found they prefer the mouth-breathing type of apparatus. After using the mouth breather a certain number of times they found it was not objectionable, although they have difficulty in talking; but talking is not necessary with a properly organized crew.

There is a general disposition on the part of people who know little of the apparatus to expect greater things of the men who wear the apparatus than they are capable of doing. The bureau thinks there should be some regulation as to the amount of work that should be required, as to the number of hours of rest for the men, as to whether or not they should take stimulants, and as to what should be the maximum distance they should proceed in poisonous gases.

B. J. MATTESON. In regard to helmet men, I believe all men that are trained to act as helmet men should pass a medical examination as to their physical ability to wear a helmet, as a man with a weak heart or affected lungs is liable to overexert himself in trying to keep up and do his share; should he fall with a helmet on in this condition there is a possibility that he would lose his life before he could be taken to the outside and properly treated.

AUSTIN KING. We train 180 men and the expense to us is $5 apiece for each practice, and that costs money.

B. J. MATTESON. We give them training at almost any point along the mine, but before we make them members of the regular crew they have to stand a medical examination. If the doctor pronounces them all right, they go on with the training; if their hearts are weak, they are generally excitable, and they might go down and lose their lives.

HENRY OWENS. For our training we have a tunnel driven 400 feet into the mountain and with only one opening. We build a wood fire right near the opening and burn a lot of sulphur. We tell the men there has been an explosion, and they are supposed to go in and investigate. They must stand a doctor's examination. In our company we train anybody.

AUSTIN KING. We train miners and drivers, and we try to give these men preference. They are the men upon whom we would have to depend. We try to get that class of men to train, and also the fire boss, who ought to make a good leader. In most cases the mine foreman is the captain of the team. Our superintendents are of no use for that work. A superintendent should stay on top; and if he is not a practical mining man, he ought to be compelled to stay there.

CHARLES ENZIAN. Do not send into the mine less than five men. At one time I was an advocate of a corps composed of few men well trained, but have since changed my opinion on this matter. It has proved better practice to have a larger number of men fairly well trained. I know of a case in point. In a party of four men one was lost because the three men could not carry him out, although he did not weigh over 150 pounds.

The colliery labor force changes very rapidly, amounting to almost a complete change, with the exception of the officials, about every three or four years. The ordinary mine workman shifts from mine to mine. We all agree with Mr. King

that the cost of $5 or more to train a man amounts to quite a sum of money at each mine, and therefore the work practically resolves itself into one classification; that is, only the salaried men are thoroughly trained, because they are more or less permanent employees.

F. W. CUNNINGHAM. In our district the mine bosses and fire bosses have to go at a 3.30 gait. Our mine bosses and mine foremen should be trained in first aid and in the use of the helmet.

At least five or six men should be on a team. It is difficult to get the right kind of men who are not afraid of work underground.

J. P. REESE. I believe in getting what you can get, using what you have. I disagree with my friend King about the superintendent. I would rely on our superintendent more than any other man on the job.

I believe that this movement is so young and there are so many problems that we will not really be able to do much at this meeting.

How far should we permit men to go with helmets? I think that is a problem we can make recommendations on. There is a time when we can keep our fresh air within a few feet of the face. But this meeting will be recognized as the highest authority on some of these problems and we can do some things.

AUSTIN KING. The reason that we do not think a superintendent is a fit person is because he knows where everything is. He is the one that has general control of everything—keeping people away, giving general instructions—and he can do more good on top than he could underground. That is why it is better to keep him out of the mine.

B. J. MATTESON. I believe that the superintendent should be trained in the use of the helmet as well as the pit bosses and other men.

J. P. REESE. I move that the joint session adjourn and that committees 1, 2, and 3 meet immediately at their respective places.

The motion was seconded and carried.

PROCEEDINGS OF COMMITTEE 1 RELATIVE TO RESCUE APPARATUS AND RESCUE TRAINING.

Committee 1 met September 23 at 3.30 p. m., adjourned at 5 p. m., and met again September 24 at 9 a. m., adjourning at noon. D. J. Price acted as secretary by appointment of the chairman of the conference and called the meeting to order. J. W. Paul was elected chairman. The members are mentioned on page 12. The resolutions adopted (see pp. 12–13) were separately presented and discussed. The discussion of importance in connection with these resolutions is reported under the proceedings of the sectional session (pp. 27–30), and in connection with the adoption of the resolutions by the conference at its closing session (pp. 51–54).

PROCEEDINGS OF COMMITTEE 2 RELATIVE TO RESCUE OPERATIONS.

G. H. Deike, acting as secretary by appointment of the chairman of the conference, called the meeting to order September 23 at 4 p. m. J. P. Reese was elected chairman. The members of the committee are mentioned on page 12.

DISCUSSION ON USE OF UNTRAINED MEN IN RESCUE WORK.

J. P. REESE. I favor using trained men, where available, in rescue operations, but where there are no trained men I favor taking the untrained men at hand.

F. W. CUNNINGHAM. I favor trained men in the use of breathing apparatus. Untrained men will probably carry out orders better than trained men in recovery work.

B. J. MATTESON. The helmet men used by our company are all trained men; the untrained men to follow up the helmet men must all be experienced mine men. The helmet men use the portable phone while advancing, and at a distance of 2,000 feet or more, considering the conditions on the inside, they put up the regular Stromberg-Carlson mine phone. A man is then stationed at the mine phone inside the mine, the helmet crew reporting to him through the small portable phone, and he in turn giving the information to the outside through the mine phone. We have had good success in following these rules; we have lost no helmet men nor had any untrained men injured.

P. F. LYNCH. The greatest dependence could be placed on the miners themselves in rescue operations. Do not allow any men to go into the mine unless they are experienced miners. With good leaders, the untrained men give good service.

WILLIAM JOHNSON. The greatest difficulty is in keeping men from rushing into the mine following a disaster. Several untrained men who are acquainted with the mines can be used to advantage.

On motion of B. J. Matteson, the chairman referred the subject, "Outline of Procedure Before and After Entering a Mine Following Explosions or Mine Fires," to a subcommittee consisting of Messrs. Matteson, Cunningham, and Raudenbush.

DISCUSSION ON MAXIMUM DISTANCE CREWS SHOULD PROCEED BEYOND FRESH AIR.

J. P. REESE. In saving life there should be no limit to distance.

F. W. CUNNINGHAM. Local conditions must govern the distance for a crew to go beyond fresh air.

J. P. REESE. I am in favor of establishing a margin of safety rather than a limit of distance. I am not in favor of establishing a limit as to number of feet to be traveled.

J. F. MEAGHER. I am in favor of a limit of time rather than of distance.

B. J. MATTESON. If birds or mice go down, the helmet men are permitted to go only 2,000 feet. If there is any possibility of there being any men alive inside the mine, the helmet squad is made up of eight men. In this case the oxygen supply governs the advance working.

On motion of F. W. Cunningham, the chairman referred the subject under discussion and the subject, "Rest Necessary for Rescue Men and Limit of Hours of Work," to a subcommittee consisting of Messrs. Lynch, German, and Zeller.

On motion of B. J. Matteson, the subject, "Cumulative Effect of Imbibing Poisonous Gases," and the subject, "Use of Stimulants," were referred back to the general conference for reference to the committee to consider the physiological effect of wearing helmets.

PROCEEDINGS OF COMMITTEE 3 RELATIVE TO SAFETY LAMPS AND ELECTRIC LAMPS.

J. T. Ryan, acting as secretary by appointment of the chairman of the conference, called the meeting to order September 23 at 3 p. m. Austin King was elected chairman. The members are mentioned on page 12.

The chairman suggested that the committee confine its discussion of lamps to their relation to mine-rescue and recovery work. He then asked for an expression from some member of the Bureau of Mines as to the practice of the bureau regarding the use of lamps in rescue work.

J. T. RYAN. The bureau's men are equipped with Wolff safety lamps, portable Hubbel electric lamps, and usually carry in addition a hand electric flash-light. In rescue and recovery work one or more safety lamps are carried by the party to test for explosive gas, except where the party is going to enter an area known to have an explosive or extinctive atmosphere.

The chairman and J. P. Bell commented on this practice and related some of their experiences at mine disasters. Both were of the opinion that the practice of carrying both types of lamp was a good one, but Mr. King did not see the advantage of carrying two kinds of electric lamps. He thought that the flash-light would be sufficient on account of its penetrating qualities. In reply to a question of H. I. Smith as to how a man would hold the flash lamp when working, Mr. King demonstrated a method of fastening the lamp on top of the cap by means of rubber bands and having the light in about the same position as the Hirsch electric lamp.

M. C. McHugh, in describing his experience in the use of lamps after mine disasters, stated that he had usually found a mixture of lamps at a mine disaster, and that he believed that electric lamps should not be used in recovery work, preferring only safety lamps.

H. H. Clark expressed his opinion on the safety of the different types of electric lamps that are in use in the mines. He briefly outlined the tests made of electric lamps by the Bureau of Mines and called attention to Technical Paper 23, "Ignition of Gas by Miniature Electric Lamps with Tungsten Filaments." He thought it impossible to obtain from an ordinary 2 or 3 cell flash-light a spark sufficient to ignite gas.

THE CHAIRMAN. Having now heard from the committee members, I suggest that we approve the practice of the Bureau of Mines and that there should be some restrictions on the promiscuous use of safety and electric lamps. In view of the fact that we are restricted in this State by law to the use of certain approved safety lamps that have been subjected to tests of the Bureau of Mines, and as the bureau is also making tests of electric lamps, I further suggest that we recommend a list of approved lamps similar to the list of permissible explosives.

J. T. RYAN. I don't know whether this committee is simply to discuss the use of safety lamps and electric lamps or not. I think, however, that we should

take some action regarding the use of open lights in mines after explosions or fires and I suggest a resolution that no open lights be used in mines during rescue or recovery work.

Considerable discussion followed, and all members were of the opinion that no open lights should be allowed in a mine after an explosion or fire. The chairman and J. P. Bell stated that this was universally understood, and asked if the bureau had the experience of being called to the scene of a mine disaster where open lights were being used. Mr. Ryan related several instances where the difficulty had been encountered, and J. W. Paul, who came in at this point, called their attention to an explosion in the Middle West that happened in a mine where gas had never been found before and where, after an explosion had occurred, three different parties went in at intervals with open lights and ignited the gas and caused an explosion each time. H. I. Smith, who had visited the mine a few days after, gave a full account of this explosion. Messrs. King, Bell, and McHugh emphasized the importance of using safety lamps in controlling mine fires in so-called nongaseous mines and pointed out the danger of using open lights. They related from personal experience instances of explosions caused by open lights in fighting mine fires.

Mr. King then pointed out the danger from open lights in a mine where the coal dust was very inflammable, and told of a mine in West Virginia where his son was mine foreman, and where two men loading coal at a working face had on three different occasions ignited dust with the flame of their torches and caused local explosions. On one occasion his son examined the place carefully for gas and found no evidence of gas on the safety lamp, and he then watched the men at work and stayed long enough to witness the ignition of the dust.

Mr. Paul stated that he had also seen the dust ignited at this mine tipple.

Mr. King had known the dust to be ignited by a torch at the tipple of another mine while coal was being prepared.

Mr. Ryan moved that the committee recommend that in mine-rescue and recovery work no open lights be used. The motion was seconded and passed without a dissenting vote.

The CHAIRMAN. I suggest putting the results of our discussion in the form of recommendations, and I shall name a subcommittee, which can meet in the morning and draft the recommendations, to be passed upon later by the entire committee. I name as members on this subcommittee Mr. Clark, of the Bureau of Mines, who is familiar with the electric lamps and has conducted tests on them; Mr. Bell, who is a Pennsylvania State mine inspector, and who is very familiar with the use of the safety lamp; and Mr. Ryan, of the Bureau of Mines, who has had experience in rescue and recovery work and in testing safety lamps.

At the adjourned meeting of the committee on September 24 all members present agreed that wherever possible both electric lamps and safety lamps should be used in rescue and recovery work and that recommendations should be made defining the limits in which they should be used by parties equipped with breathing apparatus and by those not wearing breathing apparatus.

Mr. Clark then proposed that the committee consider first electric lamps, and recommend to what extent they should be used alone, if at all, by men without breathing apparatus and also by men equipped with breathing apparatus. Mr. Clark also proposed that the committee should consider also the use of safety lamps in the same manner.

Mr. Bell thought that both electric and safety lamps should be carried by a rescue party on all occasions. Mr. Ryan objected to this recommendation, stating that oftentimes a rescue party would attempt to explore an area in which there was known to be an explosive mixture of gas or an extinctive atmosphere, or would explore an area (as a sealed section) where from the start of the exploration work there would be an explosive atmosphere. In that case there would not be any use in carrying a safety lamp, and in the former case, where an explosive atmosphere would be almost sure to be encountered, a safety lamp might be a source of danger and in any case would require careful watching, and if gas was found in dangerous quantities the lamp would have to be extinguished to proceed with safety.

Mr. Bell thought that the leader of the party should carry a safety lamp unless the conditions were such that it would be useless from the start. Otherwise the leader should carry a safety lamp and devote all his attention to his lamp; if the lamp showed the presence of gas it could then be extinguished.

Mr. Lewis thought that this was a good suggestion, and stated that this had been about the practice carried out at several disasters in which he had assisted in the rescue work.

The committee then adjourned.

PROCEEDINGS OF COMMITTEE 4 RELATIVE TO FIRST-AID METHODS.

E. B. Sutton, acting as secretary by appointment by the chairman of the conference, called the meeting to order September 23, at 3 p. m. Dr. W. S. Rountree was elected chairman.

Dr. J. W. KENNIHAN. We use in our first-aid work the Sylvester method except for broken arms, also except in cases of severe injuries to the chest.

Dr. J. W. PARSHALL. We use the Sylvester method.

Dr. R. F. MCHENRY. Regarding the Sylvester method, this is the best method of artificial respiration, provided there is no injury to interfere. It is my opinion that it is.

Dr. M. J. SHIELDS. In Europe, I understand, and some mines in this country the Schaefer method is preferred. Personally I am partial to the Sylvester method, as is also Maj. Charles Lynch, who is head of the Red Cross department I represent. The one chief objection to the Sylvester method, it seems to me, is the difficulty of getting out the tongue and holding it out if there is only one man using the Sylvester method. This feature is troublesome. I have a way of getting out the tongue that I think can be handled by one man successfully. It consists of passing a cloth around the tongue and under the chin and tying it back of the neck.

Of course, in the Schaefer method you do not have to hold out the tongue, but I do not think that you get the atmospheric pressure as directly that way as you do when the man is lying on his back. In the first national mine-safety demonstration a year ago we had an event in which a man was supposed to be burned on the face by electricity, and it was suggested that the Schaefer method be used on account of the back burns. I noted a number of the subjects when they got up had their mouths full of mud, showing that their faces had been buried in it. You can readily see that you could not get oxygen into the lungs with the mouth full of dirt.

W. D. ROBERTS. I suggest using this method: Place the center of the bandage on the tongue, pass the bandage around and cross under the chin, then pass these ends up in front of the ears and tie on top of the head.

Dr. M. J. SHIELDS. Tying a bandage around the neck pulls the jaw down.

Dr. A. F. KNOEFEL. The jaw can be held as in giving an anesthetic.

Dr. J. W. KENNIHAN. I would say with Dr. Knoefel that in giving an anesthetic a man never ties the tongue; and if he holds the jaw the patient will never swallow the tongue.

Dr. M. J. SHIELDS. He could not pump the arms and hold the jaw at the same time.

Dr. R. F. MCHENRY. Would it do any harm to the patient in case of constriction of the muscles?

Dr. J. W. KENNIHAN. I do not think you have the constriction long enough. If your man starts to breathe you are through.

W. D. ROBERTS. Suppose a man works several hours on a patient.

Dr. M. J. SHIELDS. Artificial resuscitation has been used with a patient two hours. [Mr. Shields then read Dr. Wingwright's letter stating that four men had been resuscitated by the Sylvester method, the tongue being tied.] In recommending the adoption of a preferred method it might be provided that in contests either the Sylvester or the Schaefer method, if used correctly, should receive the same merit.

R. B. Moss. To straighten out these differences some standard method should be adopted which will be given to the miners all over the United States, so that there will be one method agreed upon as a basis of judging.

Dr. A. F. KNOEFEL. We as a committee decide upon and introduce a resolution to the joint assembly, and the joint assembly either accepts or rejects it. If we recommend a method for producing artificial respiration, they can either adopt or reject it.

Dr. M. J. SHIELDS. The anthracite companies north of Wilkes-Barre agree that the correct use of either the roller or the triangular bandage should be given the same credit. Still there is a point, I think, that the triangular bandage is more practical. Of course the time would be an element. I think that a dislocation of the shoulder should be treated with a triangular bandage and that the dislocation of hips should be treated with splints.

Dr. A. F. KNOEFEL. In a dislocation of the shoulder, no matter whether forward, downward, backward, or upward, the first-aid man should put the shoul-

der in the position in which it gives the patient the least pain. I suggest padding underneath with waste, clothes, or anything, and then applying the triangular bandage as a sling to hold it in best possible position. As for dislocation of the hip, I think it wrong for the first-aid man to attempt to reduce the dislocation, unless he knows that he can not get a doctor within 6 or 8 hours.

Dr. R. F. McHENRY. The injuries should be fixed on the line of deformity. We teach our first-aid men to take the weight off the dislocated shoulder and to support it by bandaging. A splint could not always be put on a dislocated hip.

The CHAIRMAN. We have quite a number of dislocations of the shoulder and hip and we never teach our men to reduce a dislocation because he will bring more pressure to bear and in some manner often rupture ligaments. We teach our men to take their jumpers and make a pad, and the bandage is brought around the body and tied in such a manner that the patient can not move his shoulder. In dislocation of the hip joint, we put the patient in any convenient position, and the first-aid men are taught to place that limb, whatever position the patient may be in. They are also taught to place the splint from the back down to the heel, and to use a bandage in securing the splint, as by so doing the patient can be transported many miles without further injury. The main point is to make him as comfortable as possible.

Dr. J. W. PARSHALL. It seems to be the consensus of the meeting that a dislocated leg or arm should be put in the position most comfortable to the patient.

Dr. R. F. McHENRY. The points to be remembered in treating a broken back are, first, to take off the weight of the upper extremities; second, to prevent the dragging of the lower extremities; and, third, to fix the spinal column so that there is no motion in transportation. We use an improvised stretcher about 6½ feet long, with a space between parallel pieces of about 5 inches.

Dr. M. J. SHIELDS. The apparatus that Mr. McHenry presented at Greensburg on August 17 was one of the most practical things I had ever seen.

JESSE HENSON. It had been the practice of first-aid miners of the Bureau of Mines in treating a broken back to use two long splints and lay the patient straight on his back, but while in Rock Springs I learned through a physician there of a man who had broken his back and who had a kink in it. Had their treatment been used there the man would probably have died, as the physician pointed out that when a man is found with a broken back you should put him in the position you find him, as with a dislocated hip or shoulder.

Dr. A. F. KNOEFEL. Straightening a patient out is in some instances likely to cause a most serious injury. The first-aid men should merely pad the back, supporting the man in the position in which he is found, and wait for a physician, and let him take the responsibility of moving the patient.

Dr. R. F. McHENRY. The object of this splint is that it will with care meet the majority of cases of broken backs. A first-aid man would be very foolish if, on finding a man in a semiprone position, he should straighten him out. In such a case it is best to fix the body or spine in the line of deformity. In the majority of cases, however, broken backs are due to falls of rock, and should the man be found straight splints can be used.

The CHAIRMAN. My company has a number of cases of broken backs and they are handled with the patient left in the line of deformity. The company has long since adopted this method of splint, and the Bureau of Mines has the same thing, namely, two parallel bars with two pieces across them, and it seems to be an ideal splint for an injury when the man is stretched out.

ATHERTON BOWEN. I would suggest just placing a patient with a broken back on a padded stretcher, possibly securing him to it with bandages, and taking him to the hospital without splints.

Dr. M. J. SHIELDS. I recommend taking two ordinary mine blankets and rolling them up rather tightly, laying them parallel on the stretcher, so that the blankets are directly under the hollow of the injured man's back; that would take the pressure off the spine.

C. O. ROBERTS. There has been some discussion as to whether it is proper to lift a man with a broken back onto the stretcher or roll him onto it. I have always advocated lifting him, placing the hands as near to the fracture as you can. If you roll him, one side would be apt to roll more than the other.

ATHERTON BOWEN. I have personally never seen a case of broken back, but have seen several methods in first-aid practice. One was to take an ordinary mine blanket and carefully work it up under the body by moving it slightly back and forth. Another was, if the man had a coat on, to roll the coat up from the sides and pass two triangular bandages, each folded twice, under the legs; then by means of the coat and bandages lift the patient and place him on the splints or stretcher.

Dr. M. J. SHIELDS. It would depend on the position in which the man was found.

Dr. R. F. MCHENRY. I am in favor of lifting a man. I think it very serious to roll a man with a broken back.

Dr. A. F. KNOEFEL. There are two ways in which this injury may be received, either by slate falling from the roof, bending him down and doubling him up like a jackknife, or by a direct fall on the spinal column. No matter in what position the patient is found I think it better to lift him than to roll him.

The CHAIRMAN. In some cases in our mines they rolled the man on to the stretcher.

C. O. ROBERTS. The Bureau of Mines uses three men on one side and an extra man. One man places his hands above the fracture, one below the fracture, and the other man raises the man's feet.

Dr. J. H. YOUNG. The rule we use in regard to the length of the splint to be applied to limbs is to take splints long enough to go above the joint above the fracture.

Dr. A. F. KNOEFEL. In the treatment of all fractures of long bones, it is necessary to splint at the immediate fracture, including the joint above and below, so that you get a complete fixation of the bone or bones involved. If there is a fracture of the tibia, splints long enough to include the ankle and the knee should be applied. If I have to treat a fracture of the forearm, I find that by applying a right-angle splint the patient can be transported better.

Dr. J. W. KENNIHAN. I think it better in case of simple forearm fracture not to splint it at all, but merely to bandage it.

C. O. ROBERTS. I do not know whether that would be practical in mines, on account of the narrowness of some of the entries and on account of difficulty of transportation.

Dr. M. J. SHIELDS. In case of break between wrist and elbow it is the policy of the Lackawanna Railroad merely to apply a splint on the outside from the fingers to the elbow.

Dr. R. F. MCHENRY. In putting a long splint on the forearm it is possible to create a compound fracture out of a simple fracture. I thought the question referred merely to the fracture of the leg.

Dr. M. J. SHIELDS. Would it not simplify the matter if we recommend that splints for the thigh be 4½ feet long, 4 inches wide, and ¼ inch thick, and that splints for the lower part of the leg be about 30 inches long, 4½ inches wide, and ¼ inch thick?

T. B. DILTS. I move that this question apply only to the lower limbs, and that if anyone wants to make a new motion in regard to bandaging the arm he should do so.

The motion was seconded and passed.

Dr. R. F. MCHENRY. I am very much in favor of splints, as laid down by Dr. Shields in his book, for arm and forearm. It suits my work better than anything else.

Dr. M. J. SHIELDS. I do not believe in spending too much time in making diagnosis. Give the man the benefit of the doubt and put on splints.

Dr. R. F. MCHENRY. First-aid men should be taught the application of roller bandages, because there are dressings that can be done with roller bandages that can not be done with triangular bandages. I think the question should be discussed fully; that this committee should be very careful in going on record, and should avoid misunderstandings or wrong deductions.

Dr. A. F. KNOEFEL. The resolution says "in preference," and I still say that it should be used in preference. That does not eliminate the roller bandage. The roller bandage will not stay in position unless applied by some one proficient in its use. It is often wrapped too tight when, like a string tied tightly around a man's finger, it is a hindrance to circulation. The triangular bandage, although a less sightly dressing, is easily applied so it will stick.

Dr. M. J. SHIELDS. The men generally tie the roller bandage too tight in order to make it stick

Dr. J. W. KENNIHAN. I think that first-aid men should not wash a wound.

Dr. M. J. SHIELDS. Usually the man has no facilities for getting sterile water, and he will likely put in more dirt on account of his hands being dirty than he would get out. I think that we should discourage the idea of so extensive a use of peroxide. I think if you are going to clean a wound that gasoline is a good deal better than peroxide and it is antiseptic.

Dr. W. D. RICHARDS. We never teach men to wash the wound. If left alone the blood will coagulate and will form a protective cover in a way. In case foreign bodies enter and are very large, he may push them out of the way, provided he does not touch the wound. When the patient reaches the hospital, we use gasoline with about 5 or 10 per cent of iodine, which makes a most excellent antiseptic to use in those injuries.

Dr. R. F. MCHENRY. Just one word about peroxide. I think particular attention should be given to this. The blood clot is a protection and when you apply peroxide you dissolve the blood clot and open the wound, which makes it more liable to infection.

Dr. J. H. YOUNG. I think this committee ought to go on record strongly against the use of applying a chew of tobacco to a wound. [Cited a case of a woman receiving erysipelas from her son.]

Dr. W. D. RICHARDS. I move that we condemn the application to a wound of any foreign substance, other than a sterile substance.

The CHAIRMAN. In Dr. Shields's book, which we referred to a while ago, there are two opposite statements. In one place he states you must move head first, and in another case in similar circumstances he states you must move feet first. I believe that a man ought to be carried feet first.

Dr. R. F. MCHENRY. If you are carrying a man who is suffering from shock and who has a tendency to faint, you want to carry him so as to get as much blood to his head as possible. If he had an injury in the head you should do exactly the opposite.

R. B. MOSS. It seems that there has been some discussion in our part of the State about what to do first for a man suffering from electric contact,

and most instructors have been teaching first-aid men to insulate themselves before trying to take care of the patient.

ATHERTON BOWEN. I think that the current should be either cut off or short-circuited first, if possible; if this is not possible, then insulate yourself and remove the patient from the body that carries the current, or remove the body that carries the current from the patient.

C. O. ROBERTS. After you have rescued the man from the wire, is it proper to dress the wound or try to save the man's life? I claim that the most important thing to do is to revive him, get him back to consciousness.

Dr. R. F. MCHENRY. One of the main points is to lay something between the patient's wound and the ground.

T. B. DILTS. We are considering the case of a man who has been shocked with electricity so that he is insensible. The question is whether we are to take care of the burn first or revive him first.

The CHAIRMAN. In case a man gets his arm lacerated in a machine, the first thing to do is to put your finger over the artery or use a tourniquet, which stops the flow of blood, and in this manner you will be able to save his life. I therefore believe that in electric shock, when a man is insensible, you should first attempt to revive him by artificial respiration before dressing the wound.

Dr. R. F. MCHENRY. I do not advocate wasting time with dressing a wound. By the time the man is removed from the wire somebody has a pad underneath him.

ATHERTON BOWEN. From which side should the first-aid corps handle the patient? For instance, in a compound fracture of the thigh, after splinting him, should the men work on the injured or uninjured side in putting him on the stretcher?

Dr. J. W. KENNIHAN. The first-aid men should be on the uninjured side.

W. D. ROBERTS. If a man is injured on the right side he should be handled by the corps on that side, because when they raise a man in placing him on the stretcher the side away from the men lifting will fall a little. I say that the injured side should be next to the men lifting the patient.

R. B. MOSS. The American Red Cross recommends that the patient be handled next to the wound and that the stretcher be placed closest to the wounded part.

The CHAIRMAN. If this organization goes through there ought to be uniformity in the methods used in the United States; when Indiana meets Pennsylvania, when Tennessee meets Colorado, they should all use the same methods of dressing.

Dr. R. F. MCHENRY. My idea is to allow the first-aid men after any contest to demonstrate any new devices they may have developed, and to have these devices passed on by a board of censors and accepted or rejected. This course will stop taking on first-aid features that are practically experimental.

The CHAIRMAN. I recommend that an organization be formed and that this organization be the arbitrators of dressings to be used in first-aid contests.

R. B. MOSS. I move that if this organization is perfected we recommend a committee of seven persons, consisting of two first-aid men, two operators, two physicians, and one representative of the Bureau of Mines, to act as an advisory board or executive board, that any time any new dressings are brought forth at these contests they be turned over to this advisory board for an opinion as to whether or not they should be used by first-aid men.

PROCEEDINGS OF JOINT MEETING OF COMMITTEES 5, 6, AND 7.

J. J. Rutledge, acting as secretary by appointment of the chairman of the conference, called the meeting to order September 24 at 9.40

a. m. Dr. M. J. Shields was elected chairman. The members are mentioned on page 12.

Dr. G. H. Halberstadt discussed the death rate before and since the establishment of first-aid work, stating, among other things, that in 1911 11,000,000 tons of coal was mined, and that there were 82 deaths—1 death to 144,000 tons; that the Philadelphia & Reading Railroad Co. was the first company to take up the organization of mine first-aid corps, starting with 400 men; that carron oil is no longer used for dressing burns, gauze saturated with a 2 per cent solution of picric acid being at present used; that the dressing often remains on for 48 hours.

Dr. Halberstadt cited the case of a man who had been injured in a railroad yard. When the patient, who had been meanwhile treated by a doctor, arrived at the hospital Dr. Halberstadt found it was not possible to operate on him even after working upon him for 48 hours. Identically the same accident happened at a mine and was treated by a first-aid man. There was no loss of blood and the patient was successfully operated upon at the hospital, as the shock had been properly treated previous to reaching the hospital.

Dr. Halberstadt said further that the trouble with doctors is that they do not have the appliances at hand to treat such an injury, or they think it is a case to be treated in transit, and so handle it. He stated that in the first-aid work in his field the appliances are well placed and the first-aid work well done; that treatment is free and that indemnity is paid during certain loss of time.

Dr. J. W. PARSHALL. The Frick Company makes payments to men while in the hospital; also so much indemnity for the loss of an eye, a limb, etc.

Dr. G. H. HALBERSTADT. We keep them on relief for six months, and the company has been paying to this fund. We have three State hospitals and one local hospital that take care of the injured men. I see the men only in case they are very seriously injured, when I may be sent for to look after the case.

The CHAIRMAN. I think that we have two things to discuss here: The economic value to the company and to the men of first-aid training, and then the topic of mine hospitals, in regard to where they should be located, etc. I can indorse what Dr. Halberstadt has said, as it has been my experience in the upper region. I have a number of hospital reports of the hospitals in Scranton; also, I want Dr. Rountree, of the Tennessee Coal, Iron & R. R. Co. to quote figures that were given out by them about two years ago. I assisted Dr. Rountree and other surgeons in organizing first-aid teams in 1909, when the death rate was about one man for every 90,000 tons of coal mined. The following year it dropped to one man in every 140,000 tons, and the following year it dropped to one man for every 190,000 tons of coal mined. You can not attribute all of that to first aid, but first aid played an important part. Perhaps 80 per cent of the accidents that happen have an element of contributory negligence. The mere agitation regarding accidents, talking about them, the introduction of first aid, gets the men to thinking about them, makes the men more careful for themselves, and they warn others, and also gets the company

interested in making the mines safer, and the two cooperating with each other cut the death rate down.

Dr. F. L. McKee. Previous to using first-aid corps it has been difficult to get men who have been hurt to report to the mine hospital for treatment. Since the order went into effect to report to hospital for treatment, there are only about half the number of men drawing fees from the relief fund that did before, showing that many men feigned illness. In connection with dressing wounds, we wash all our wounds, using 4-ounce hard-rubber syringes and a 2 per cent solution of soluble cresol. The only two cases of infection that we had were cases that did not report to the hospital to have their wounds washed.

Dr. W. S. Rountree. My experience has been somewhat that of Dr. Halberstadt. I will say that in the past few years since we have first aid in some of our mines, our death rate has been cut down very, very much, far more than was ever expected, and the percentage of infection lower than ever dreamed of. The wounds of our men are dressed when they come from the mine, and I have seen only few infections in two or three years where the first-aid men get hold of the men soon after the injury. We do not wash wounds. We do not allow our first-aid men to wash them. If a first-aid man hasn't a first-aid box near him when an injury occurs, he simply leaves the wound open, bare, rather than to cover it with a piece of his jumper or other material used in the mine, such as waste or brattice cloth. They don't put tobacco on a wound any more, and if they leave the wound open, nature soon forms a dressing with blood clots and it seems to act beneficially. When we surgeons get hold of a patient we don't probe, don't put our hands in the wounds, never attempt to get inside. In other words, if the wound is covered with dust and dirt we use about a 5 per cent solution of gasoline and iodine to clean it, and then we put on sterile dressing. When the wounds are treated in this way we very seldom have infection, and as a result the wounds seem to heal quickly, and it is not very long before the man is back at his work.

There are a number of things that we teach the first-aid man to use for burns. I like a dressing made of a saturated solution of bicarbonate of soda, it seems to act beautifully. In first and second degree burns the first-aid man puts on a sterile dressing saturated with a solution of boracic acid or bicarbonate of soda. For third-degree burns we use the old fashioned balsam, which makes a very nice dressing, but is somewhat painful.

I believe that it is a wrong idea to allow first-aid men to wash a wound. Dr. Murphy, of Chicago, a surgeon of wide experience, says that he finds it best to never probe or fuss with a wound, that his success in treating injuries has been remarkable, and that this view is confirmed by other experienced surgeons.

Concerning electric shock: When men suffer little or no shock the first-aid man is taught to give aromatic spirits of ammonia. In the long-ago, before first aid was introduced, whisky was a popular remedy and a very poor one, as it has a tendency to increase shock. The percentage of deaths has been cut down very greatly in the past five years, so that it is no doubt of great economic value to the companies to have first aid in their mines, mills, etc., and certainly it is of great benefit and help to the surgeon in charge. When we can get hold of a man who has been injured and later dressed by first-aid man we feel pretty sure that the fellow is going to return to his work soon. However, no matter how beautifully a man is dressed by his fellow workman it is the surgeon's duty to open the dressing and inspect the wound, so that if anything has been done wrong he may be able to correct it before any damage has been done.

R. B. Moss. I would like to ask Dr. Rountree whether or not they find it advisable to wash a wound that has been caused by a man going back on a shot. In the Kentucky coal fields we have quite a bit of solid shooting coal, and such accidents are common. Getting pitted with coal and dirt leaves a horrible-looking face on a man. I would like to hear this matter discussed, whether it would be advisable to wash such wounds or not.

J. J. RUTLEDGE. I was given the circumstances of such an accident by a superintendent of a big coal company in the State of Washington. The general manager, the superintendent, and some of the foremen went into the mine and walked up on a shot. The first man in the party was killed, the others injured. The man most severely injured kindly allowed us to examine him, and there were 107 punctures of the skin, and in some of them I could see particles through the skin as large as the end of my thumb. He said he wasn't washed at all. The man had absolutely perfect use of his limbs, his functions seemed to be all right, and his condition splendid. It was a remarkable accident and an exception.

Dr. W. S. ROUNTREE. We have quite a number of injuries in our mining district of this sort. Long ago men whose faces were blackened by explosions went through life that way. They were simply treated in a way, and the coal dust remained in the wounds. In the last few years we have never sent a man away with a black face after such an accident. With a salt solution and a stiff brush every particle of coal dust may be removed. Sometimes the coal dust is buried so deep it can not be reached with the stiff brush, but with a curette all particles may be removed. The only way we have been successful with these patients was to immediately anesthetize, scrub with brush or curette, wash with salt solution, and dress with sterile bicarbonate or some other dressing. Sometimes after a patient goes 24 hours or so the face swells, and many particles are shoved up, and are thus much easier to remove, even at that time; but by letting them go so long more infection is caused. However, such infection is not so dangerous as infection received on the surface. The use of a stiff brush destroys some tissue, but causes no scars, and practically all the dirt can be reached with the brush.

Dr. J. W. PARSHALL. I perfectly agree with Dr. Rountree; but it appears to me that we are getting into a medical discussion of surgeons instead of first aid, and I think we had better stick to the text.

With regard to the economic value to the company and to the men of first-aid training in hospitals, in my experience it has been very economical to the company where the company pays the bills and also has the services of the man before it would have had them if these wounds had become infected by reason of not receiving first-aid treatment. The greatest trouble is to get our men to report to the first-aid man, and I have had such cases as a man accidentally striking himself with a pick and neglecting the wound so that a stay of four months or more in the hospital was necessary, where a few days would have sufficed if he had received first-aid treatment. We urge our superintendents to have first aid administered, and every time one of these men becomes infected we report it, and the head offices get after the superintendents to warn the men and try to give them instructions so when a wound occurs, no matter how slight, they will have it treated by a first-aid man and then reported to the surgeon. Neglect causes most of the serious injuries and creates loss of time and men.

Dr. R. F. MCHENRY. I do not know that I can add much to what has been said regarding the economic value of hospitals, as we are too young in the business. In about seven years' experience in an isolated mining town in western Pennsylvania, starting with crude hospital equipment, we have had our troubles.

As far as economic value to itself is concerned the company is very well satisfied with it. We started with an assessment of 10 cents per month for injuries only, and we raised it to 20 cents and finally to 40 cents; this assessment covered not only accidents but all sickness. The company built a hospital at an expense of about $20,000 and we furnished it and are running it without State aid. We care for 15 patients per month. Also, we look after sanitary conditions. We make it a point to prevent all the sickness possible. The company has a man going over the town with a garbage wagon collecting all garbage, disinfecting cesspools, etc., and we have had only two cases of typhoid fever in seven years, with a population of approximately 3,000 people, and we have had comparatively few cases of sickness of any kind.

We instruct our first-aid men and try to make missionaries out of them. We tell them that no matter how trifling an injury is the injured man should be sent to the physician, and I want to say that since the work was organized it has been a rare thing for a man to come into our offices who hasn't had first-aid treatment or has failed to report himself promptly. The days of infection are pretty nearly over. The men are well taken care of in the mines, their wounds are treated properly, and recoveries are much more satisfactory and a great deal more rapid. We are mining about 3,000 tons of coal a day, and at the most we have had only one death from accident this year.

Dr. G. H. HALBERSTADT. Dr. Rountree says they never get infection from mine wounds. In the anthracite region the contrary is the case. We don't have the wounds washed, but cover them with sterile gauze. We have splendid facilities for moving patients to hospitals, and there the surgeons do as they see fit. The cases that are not disturbed immediately by surgeons are probably severe burn cases in which men are burned from the waist up; if the burns are dressed with picric-acid gauze and the patient is suffering from shock the surgeons let such cases go for 24 hours without further dressing. We have found that it is a punctured finger or a slight puncture of the foot that takes away most of the relief fund. I had a sign put up, printed in several languages, that men must report the slightest wound before leaving the collieries. I have found on Monday morning men would come back to work with injuries they had received Saturday night and didn't report. If I can stop infected fingers and feet we can increase our indemnity still further, but most of the foreign miners go home, poultice a wound for a week, and after it is infected, report to the doctor, and often a lost limb is the consequence. The requirement ought to be that all men who want indemnity must report accidents immediately and must have wounds treated properly if they want to get any of the beneficial fund.

Almost all wounds of the eyes, corneal wounds, become ulcers, followed by loss of vision. All such cases we are trying to drive to the doctors and hospitals and so reduce the time of disability. This last year I have not talked to any of our men on first aid. I have been hammering into them conservation of human life. I have submitted a new form of accident report which I wish to submit here.

As a whole, our work is coming out very well. Eighty per cent of our men are injured through their own carelessness. We don't allow men to clean wounds, because they can not do it properly, and the time is wasted. We tell them simply to cover wounds up, putting on sterile dressings or tourniquets.

J. W. PAUL. Will Mr. Moss tell the committee how his company has spent $25,000 in sanitation at their mines during the last year?

R. B. Moss. When my company purchased the 15 mines in Bell County, Ky., the sanitary conditions were very poor. There were 150 cases of typhoid

fever; several hundred cases of bowel complaint, and fully 50 per cent of the entire population of the camps was affected with hookworm. The typhoid was quickly traced to the open springs and wells. These were closed, and 25 new wells were bored, none of them less than 100 feet deep. Over 400 new closets were built, and built in such shape that it was practically impossible for a fly to get to the excrement. The rubbish, filth, etc., were hauled away and burned, and the houses were cleaned, with the astonishing result that in the summer of 1912 not a single case of typhoid appeared among the Continental Coal Corporation's 6,000 employees, and very few cases of bowel complaint were reported to the company's physicians.

The hookworm campaign was started under the direction of the Rockefeller Institute by the State board of health of Kentucky, and at this time the work is being carried on with great success, the people responding generally. The treatment is very simple, and does not keep the men from working.

All of the houses, fences, and closets were whitewashed, and all of the above-mentioned things were done, with the result that in the fiscal year ended June, 1912, my company loaded about 22 per cent more coal with the same number of men than did the old companies under separate management.

Dr. J. W. PARSHALL. We have a number of hospitals installed, and my assistants attend to the outside part of this work, and I am not very well versed in that work. I think we ought to try to recommend some action as to how to get men to report when they are hurt.

The company has spent quite a great deal of money in fitting up emergency hospitals, without their being of any use to us, for, as a rule, these patients are generally immediately sent to the outside hospital. A room with proper first-aid supplies, or possibly a table for a surgeon, if present, would be sufficient.

Other companies have reported that they could pay a benefit by assessing the men. Our company pays all expenses from the time the men are injured, pays all bills, transportation, etc., and pays 35 per cent for single men and 45 per cent for married men, with 2 per cent added for each child under age. In regard to this I think it has gone beyond what is necessary in putting in mine hospitals, because we haven't used them. Our men are all transported to the general hospital as soon as taken from the mines. In the hospitals which I represent at Uniontown, we treat probably half the patients of the company, which has 68 mines.

Dr. W. S. ROUNTREE. We are trying to work out a plan of hospitals by having an emergency hospital at the mines, and a base or central hospital somewhere in the district. When a man is injured, a first-aid man takes charge of him and treats him, and he is then sent to this emergency hospital; if suffering from shock he is treated there for such condition, and if he appears "hospital worthy" he is sent on to the central hospital. I believe an underground hospital impracticable. The men who are injured are never taken to the so-called hospitals underground. They are attended by first-aid men and removed to the surface. Consequently it doesn't seem the thing to have a hospital underground. There is but one in our section, and nothing is being done in this hospital. I believe a system of emergency hospitals with a central hospital will be the plan of the future.

R. B. Moss. Our company recently, at the request of its employees, equipped a hospital in Pineville, Ky., with 20 beds. The building and equipment is furnished by the company, the equipment being entirely modern in every respect. We have a medical staff and advisory board of several surgeons and physicians, and four trained nurses in constant attendance.

The hospital is supported by a voluntary cut upon every employee of the company, from the general manager down, of 25 cents per month. The hospital has been a success from the start, and it is my opinion that it will result in a decrease of the number of damage suits against the company. It is an emergency hospital only. Patients are accepted, however, from other mining companies upon payment of $10 per week. The cut of 25 cents per month includes all charges so far as our employees are concerned, and it is thought that in time this cut can be reduced.

A recent case in one of our mines was that of a miner's daughter, who was allowed to stay at home and be doctored for stomach trouble until she was almost dead with appendicitis. She was taken to the hospital, operated upon successfully, and left the hospital after staying there five weeks. The only expense to her parents was the 25 cents per month paid.

Dr. R. F. McHENRY. So far as underground hospital equipment for surgical operations is concerned I don't think it is the right plan. Physicians ought to refrain from asking companies to waste money in work of no benefit. I think when you are equipping underground for surgical purposes in the ordinary mine of western Pennsylvania you are throwing money away. I know of no place in our district that requires underground surgical equipment at all. Money so expended could have been used in a different way, one which would be of more benefit to the men.

In seven years' experience we have had only one man bring damage suit against the company, and he asked for only $150, which was given to him.

Dr. G. H. HALBERSTADT. It would appear that such stations or hospitals as are required by Pennsylvania mine laws are simply first-aid stations. It has never been intended that the doctor should go underground and operate.

I merely reverse conditions and take the dressing station to the patient. He is then taken to the surface and shipped to the hospital. We have dressing stations installed inside and outside of the mine. Dressings would keep sterile, but it would be impossible to keep instruments underground. A man when injured wants immediate practical assistance, and wants to have confidence in the man doing the work, and wants to see daylight. When a man's wounds are dressed at the surface he is transferred from the room to the ambulance and rushed to the hospital.

My advice is this: When a man is dressed put him in a mine wagon or a spring litter such as we have, and rush him to the slope or shaft and hustle him to the surface. A man wants to see daylight when hurt. First-aid equipment is all underground; it is taken to the man when hurt instead of the man being taken to the intermediate station.

Dr. M. J. SHIELDS. It is the opinion of this committee that underground hospitals are unnecessary and that there should be deposited in the mine a sufficient number of properly equipped first-aid emergency cases of dressings.

Dr. A. F. KNOEFEL. First-aid rooms are often used, when a man is killed, as a place for keeping the body until the undertaker arrives and where the body can be prepared for burial without trouble to the family.

Underground hospitals are unnecessary. There should be deposited at different points in the mines a sufficient number of properly equipped first-aid emergency cases, so that they can be quickly taken to any part of the mine.

In the Vandalia Coal Co.'s mines in Indiana the men carry first-aid packages, consisting of sealed metallic boxes containing a compress, triangular bandage, and safety pins. First-aid dressings should be brought to injured men, rather than to move the men to the dressing. If the man had a simple fracture, it might become a compound fracture, or soft tissues might be otherwise injured

by moving him to the first-aid station. A large first-aid cabinet should be placed at each parting.

Dr. F. L. McKee. I am going into the mines nearly every day. If first-aid stations are installed at the surface of a mine, they would have to be protected in some way in case of a serious accident, as the relatives and friends on the outside would likely tear them open to get to injured persons.

C. O. Roberts. I think a dressing room is the proper thing. In some cases it would be advisable to have a dressing room, where an injured man could be taken and kept warm until he recovered from shock or other injury.

At the mines in the anthracite region, where I was captain of a first-aid team, we would take the first-aid cabinet to the man, dress his wounds, and then take him to the dressing room and keep him warm until he could be taken to the surface.

Dr. J. W. Parshall. Regarding hospitals: The places referred to by me were fully equipped as hospitals, with operating tables, etc., and I consider it unnecessary expense. As the plants were widely distributed, there was more reason for such arrangement. First-aid stations are maintained throughout the mine, and about one first-aid man in ten is distributed around the mine, so that prompt treatment can be taken to the injured man. We have instituted a system of telephoning in the larger mines, but I think, as these other gentlemen do, that it is absolutely necessary and important to have a receiving room in case of delay or some reason why a man should be treated before being taken to the hospital. I merely criticize the too elaborate hospitals.

Austin King. In all our plants the company has taken up conditions of hygiene and sanitation, and it has spent a half million dollars making open drainage and other improvements, fixing up property, and also for policemen, that is, men who look after contagious diseases, sanitary conditions, etc. These things are all done at the expense of the company.

It is our practice to distribute first-aid supplies to the points most convenient to those where men are employed. For the safe storage of these supplies we use tubes made of galvanized sheet steel of proper gage, 8 feet long and 8 inches diameter, to contain stretcher, blankets, etc., and one of similar diameter 2½ feet long which contains first-aid supplies. These supplies are to be kept sealed so they are always dry and fit for use at any moment. Some superintendents go further and have a painted sign with the Red Cross on it, so the men know where a first-aid station is. Usually a telephone is close to each station, and when supplies are taken the first-aid man telephones to the surface for a doctor to be summoned.

Speaking of treatment: First-aid supplies are carried to the place of an accident, or to a place of safety, and treatment is given for shock. Safety lamps are placed under blankets and aromatic spirits of ammonia are given and other measures taken to keep the patient warm before bringing him to the surface.

Atherton Bowen. One point partly brought out is the sealing of first-aid cases; this is done in the anthracite regions. Material must be kept dry or it becomes absolutely unfit to use in case of accident.

H. M. Wilson. Dr. Knoefel wasn't sure whether to speak of stations, hospitals, first-aid stations, or what. It has developed here that the law of Pennsylvania requires using underground hospitals, and it appears that the companies are going to considerable expense to establish hospitals. I think one of the things this conference might well consider is the question of terminology, because our deliberations may have an important effect upon legislation. This conference should fix terminology as to what is a hospital—whether under-

ground places for rehandling should be called first-aid stations, hospitals, or what.

Dr. F. L. McKee. I think we should retain our inside stations and prepare for others outside.

G. W. Barager. The Pennsylvania laws require the use of carron oil. Patients dressed with this oil excited unfavorable comment on the part of the hospital officials.

Dr. G. H. Halberstadt. We keep carron oil, but have the corks driven in so tight they can not be removed.

J. W. Paul. I agree with Mr. Wilson's remarks, and I think the next thing in order would be to consider a motion indorsing first-aid stations or dressing rooms in the mines. A great many men suffer, especially during the winter months, when brought out in a crippled condition, whereas were a room provided, suffering would be very much relieved.

PROCEEDINGS OF COMMITTEE 5 RELATIVE TO FIRST-AID TRAINING.

J. J. Rutledge, acting as secretary by appointment of the chairman of the conference, called the meeting to order September 24 at 2 p. m. Dr. A. F. Knoefel was elected chairman.

Dr. M. J. Shields. It has not been the policy of the Red Cross to take the doctor's work away from him. The duty of a first-aid man is to form the bridge that takes the man safely over the danger of an infected wound, over the danger of unnecessary or exhausting hemorrhage, over the danger of possible crooked leg or amputation, over the danger of dying unnecessarily after a noninstantly vital electric shock, and on the other hand to place in the hands of the general hospital or the physician a man who has a cleaner wound, a straighter leg, perhaps, a man who is suffering from less shock, suffering less pain, and a man who will get well from 10 to 60 days sooner than he would if he had not had this first-aid treatment.

Dr. F. L. McKee. I think that Dr. Shields has covered almost everything that can be said on this subject. The duties of a first-aid miner will depend on his general aptitude for the work, whether he has the nerve to do the work, and whether he will obey explicitly the instructions he has received and will follow them to the letter.

Dr. G. H. Halberstadt. The first-aid miner should be educated in the work. He should get to his injured comrade as promptly as possible in order to put on proper dressings, then get the patient into the ambulance and off to the hospital. One thing I especially impress upon my men as much as anything else is to make no prognosis. If a man has a finger injured, the first-aid man should not say, "Do not let the doctor take that finger off." Their business is to cover the wound, using splints or sling or any other apparatus necessary, and to get the man into the hands of the doctor as promptly as possible.

Dr. R. F. McHenry. The question of who shall instruct the first-aid men is a rather delicate one for a physician in practice. Any man who has sufficient preliminary education to understand the subject matter of first-aid treatment and is an experienced instructor is capable of instructing first-aid men. We have both physicians and laymen instructing first-aid men, but only laymen of experience are capable of giving such instruction. The amount of first-aid training or instruction that is necessary can not be limited by hours or days or time. In the method of obtaining first-aid training and instruction what every individual needs is to get the work and get it in a way that he can apply it. To

accomplish this it is necessary in the first place to teach him the subject matter, and in the second place to do the actual work on a well subject, until the first-aid student becomes skilled in the actual work.

The CHAIRMAN. A superintendent was outlining last night the plan adopted by his company, whereby the man, after taking a certain amount of first-aid and rescue work, was issued a certificate of proficiency. If he went from Indiana to Pennsylvania he could present that certificate and receive a certain amount of recognition. The question is one that is proper and just. Here is Dr. Young, who requires his men to meet once a week, two hours a night, for three months; Dr. Rountree, who has his men meet every other night for two hours during six or nine months. There ought to be some standard to go by, and I would like to hear a thorough discussion of this question by all of you as to your personal opinion of what you believe should be the standard.

Dr. G. H. HALBERSTADT. I do not know anything about the Red Cross system of training. In the first place, we ask for volunteers. When the whole thing was started I showed the men first the simple bandages, avoiding technical terms, giving them simple injuries—for instance, a broken leg, by simply demonstrating from binding a broken shaft. From treating broken legs I went on to treating hemorrhages. We have charts of the human body which we study, and I have taken the men on and on, at the same time increasing the difficulty of the bandages. Now, you can not take the same set of men and keep them interested in working all the time, so I go around and talk to them on anything but the application of bandages. You have to give them some other entertainment and get on some other subjects. I have been talking to them largely this year on the conservation of human life. Each year we have an annual contest which is practically on the application of bandages.

Dr. W. S. ROUNTREE. I have been teaching first aid for a number of years, and our methods have been similar to those of Dr. Halberstadt. In organizing our corps we first ask for volunteers or select so many men from the works or the mines, and after fixing meeting nights and getting classes arranged I give about 20 or 30 minutes' talk on anatomy and physiology, omitting all technical terms, simply using as plain words as I can to express myself.

After my lecture I demonstrate from the chart and then I take up simple injuries; for instance, an injury of the hand, and teach the men how to dress it. We go from that to more severe wounds. We will take up a simple fracture of the forearm and spend one night teaching the applications of bandages and adjusting splints, then compound fractures, and so on throughout the whole calendar of fractures until we have accomplished the treatment of that class of injuries. I teach the men never to reduce a dislocation.

Occasionally we have a night devoted to a quiz. I mark the men according to the answers and keep a record of the marks. At the end of the term we take well-informed men and put them into a separate class and later give them a corps to instruct, and in this way we have been able to keep up interest. I will also say that other things must be injected in this work. We have little smokers, banquets, musicals, and some nights are given over to talks on how to keep up physical efficiency and to prolong earning capacity. I also touch upon the social evil, as miners are prone to fall a prey to such diseases, and by these methods I have been able to keep the men interested in this great work.

Dr. F. L. MCKEE. I have been introducing Morrow's book, L. H. Doty's book on first aid, and the Red Cross book, and recommend each of them. So far as organization goes we watch a man for his aptitude, and get somebody else if a man does not show a general aptitude for the work.

T. B. DILTS. We have invariably adopted the industrial edition of the American Red Cross and we have asked the doctors who have charge of these differ-

ent classes not to spend too much time at the beginning on lectures, but just as quickly as possible after the second or third lesson to start on practical work—such as the use of different bandages, splints, etc.—and wherever they have done so the classes have been successful; where they have not, the classes have generally broken up after the seventh or eighth meeting and it was hard to interest the men again.

Dr. R. F. McHENRY. My experience has been small, but as a matter of fact I think we must have standard instructors for our men. I read the book to the men and at the same time I have the charts. I then ask the men to stand up and answer questions. This method makes them confident and relieves embarrassment. At the end of the lesson a short time is given to anatomy and chemistry. We spend about three months on that first practice before the final examination. The Red Cross send their representatives to examine our men and we have had only one man fall below 75 and only four men below 90. We need a standard book that is generally before the men employed in and around the mines. I personally think that the Red Cross book is more generally before these men to-day than any other book we have. It is not a finished book; I do not presume it was intended to be. It should be revised; at least, there are many things necessary in first-aid instruction that are not emphasized in this book.

JESSE HENSON. We follow the American Red Cross textbook, industrial edition, in the Bureau of Mines. We go into a mining camp about a week at a time. We use the most simple methods possible in training, and in tying bandages, etc. In some places where we have organized first-aid classes the miners furnished their material, each man paying 10 cents per month in some places and in others 25 cents per month; in still other places the men held social functions. But to continue the work we always had to look to the physician.

The CHAIRMAN. At the beginning of the first-aid movement we had two classes a week for two hours, from 7 to 9 p. m., on Tuesday and Saturday. As the disinterested dropped out we combined the meetings to once a week and this class took nine months to finish. The book we adopted and the one we still use is the American Red Cross textbook, industrial edition.

We use both roller and triangular bandages, with preference for the triangular. We first started on anatomy and hygiene, and followed these subjects with physiology.

The first-aid men to be competent must know that cleanliness is far beyond godliness in first-aid work.

Dr. M. J. SHIELDS. The Red Cross Society, one of the greatest societies the world has ever known, is not confined to the United States. We desire to receive any suggestions. We want to cooperate with any movement that has for its object the benefit of humanity, the same as we are cooperating with the Bureau of Mines, the Y. M. C. A., the Y. W. C. A., the Boy Scouts, and the Boys' Brigade. We have, through the assistance of the Surgeon General, promulgated what will be known as the medallion course; that is, the men will bear standards and will be somewhat similar to the National Guard. They have to be examined physically, etc.

Maj. Charles Lynch was the author of the Red Cross industrial book, which was based in part on the first-aid book of the St. John's Ambulance and a little book I had published myself. The Red Cross general edition, whose author is also Maj. Charles Lynch, is used by the Y. M. C. A., the Boy Scouts, and the Y. W. C. A. The Red Cross publishes the industrial edition in four foreign languages. We would like to get suggestions in regard to revising the book.

We are arranging a pamphlet of lectures for railroad men. Sometimes the railroad men can not attend because they are at the other end of the line. By the use of these pamphlets they can study what the class is being taught while they are absent. I might say that about 25,000 copies of the industrial edition have been sold throughout the United States.

Dr. G. H. HALBERSTADT. I think a good method of organizing a first-aid corps is to take five boys out of a colliery and show them the work for three hours, one hour at a time, for three weeks. I use X-ray pictures and motion slides as much as possible, because this interests the men. If you want to keep your corps together to do good work, to do work that you will not be ashamed of, to do work that will stand out, keep them on the roller bandage; give them something to fight for.

The CHAIRMAN. When my company began the organization of the first-aid association, the different locals refused to enter into the movement, although the company agreed to pay their expenses; but later an independent first-aid company was organized and the members of this association included laborers, loaders, and anyone who desired to take up the work. The entrance fee of this organization was 50 cents, each member being presented with a Red Cross first-aid textbook and a metallic first-aid package. The dues were 10 cents per month, these to be used for incidental expenses.

Dr. D. H. LAKE. I would like to say how I made a failure. I got interested in first aid and bought charts and first-aid books and had plenty of bandages. Called a meeting and about 50 men were present; this was the first meeting; gave a lecture probably too long, but I thought I was making it interesting, a talk of two hours. Got out all the bandages, distributed them, went around and showed them how to apply them. Arranged for a meeting the next week; there were about 20 men present at the next meeting. We went over the bandages again; all were very much interested. The next meeting about five men appeared. I tried to find out why they did not appear. Some men seemed to belong to societies, etc., and had to go to them the night we were to meet. We tried to have a night that would not interfere. The next meeting I was the only man there. I am here to find how you keep the men interested in the work. I will start another first-aid corps if I can find that out.

Dr. M. J. SHIELDS. It has been my experience in a number of years in this work that where you find the officials passive you find the men passive; and where you find the officials active and interested in this work you find the men interested, and it will be very easy to get them. In the last two years in our Red Cross work we have covered about 75,000 miles of railroads, including the New York Central, Rock Island, Illinois Central, Harriman lines, Santa Fe, Northern Pacific, Milwaukee, Baltimore & Ohio, and Erie, and when we get to a division where the superintendent of that division is heartily interested we find the men get interested. If the superintendent thinks that it is something extra to the regular work, we do not get any results.

T. B. DILTS. In one place where we tried to interest men but had not been successful, I went over and got six of the men together and told them that was all we wanted. After about an hour's demonstration they got very much interested and arranged to meet one day a week to practice under the direction of the superintendent. After about three months they were each asked to captain a team. We have 35 men there to-day organized and greatly interested in first aid. At another place we had the same thing. We started with five men and after about a year's training of these men each one was then put in as captain of a squad.

The CHAIRMAN. What is the opinion of the men present as to the ratio of trained men per 100 miners?

Dr. M. J. SHIELDS. Of the coal-mining companies whose work I have helped start, the Hillside Coal & Iron Co., the Temple Iron Co., and the Pennsylvania Coal Co. tried to get at least 1 man in 20. The Tennessee Coal, Iron & Railroad Co., 1 in 25, the H. C. Frick Coke Co., the same, the Oliver Iron Mining Co., 1 in 30. We contend that it is a good theory to ask for more men than you want, so that if you do not get the allotted number you may get a sufficient number. When with the H. C. Frick Co., we planned to get 1 man out of 50. We thought this would be all we could get together. We held some of the meetings in the daytime. We got together 750 men out of 15,000 employees, and our last meetings were larger than the first. The people were invited to come whether they were asked by the foreman or not. All these men were asked by the foreman to be present.

The chairman then asked each man what was the proportion of first-aid men to the number of employees in the companies that they represented. Dr. Rountree said 1 man to 10, Dr. Halberstadt, 1 man to 70; Dr. Young, 1 man to 80; C. G. Brehm, 2 men to 25; Dr. Knoefel, 1 man to 50; Dr. McKee, 1 man to 25; E. E. Judd, 1 man to 20.

CLOSING SESSION OF CONFERENCE.[a]

The closing session of the conference was called to order at 10 a. m., September 25, 1912, by the chairman.

The CHAIRMAN. The several committees have met and completed their work. The action of each committee has been referred to the resolutions committee, the membership of which is as follows: Dr. F. L. McKee, chairman; Messrs. B. J. Matteson and A. M. Ogle, Dr. W. S. Rountree, and Mr. J. W. Paul, secretary. Will Dr. McKee report for the resolutions committee?

Dr. F. L. MCKEE. The committee on resolutions has met and has reviewed the reports of all committees and transmits them to this general session with the recommendation that they be passed upon separately.

On motion of J. P. Reese it was agreed that the report of the resolutions committee be adopted, with the understanding that in the reading of committee reports if no objection was raised to any section the chair was to declare it adopted to avoid debate on each question.

The chairman then directed the general secretary to proceed with the reading of the resolutions recommended for adoption.

Only these resolutions are reported and entered in the record at this point which were discussed or amended. The others are recorded as originally drafted. (See pp. 12–17.)

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 1, RESCUE APPARATUS AND RESCUE TRAINING.

J. P. Reese asked a definition of the word "station" in resolution 3, which read as follows:

"The committee recommends that all mine rescue stations should be equipped with at least five breathing apparatus and the necessary

[a] In connection with the report of this session frequent reference is made to the resolutions presented on pp. 12–17

accessories for the continuous operation of the apparatus for 24 hours, and at remote stations 48 hours."

J. W. PAUL. A rescue station is one equipped with certain paraphernalia and apparatus suitable for mine-rescue work and mine-recovery work, and the committee recommends what this equipment should be for a regular rescue station.

J. P. REESE. The establishment of rescue stations is voluntary on the part of any company, and we do not want to discourage the establishment of stations with even two rescue apparatus. If a company has several mines within a radius of a few miles, we do not want to discourage the establishment of a rescue station at each mine, even if it is equipped with only two helmets, providing the company has a car or a central station or additional small stations from which to draw in case of accident. The company does not wish to go to the great expense of having a fully equipped rescue station at each of its mines, and I think the specification of five helmets should not apply to substations.

G. S. RICE. We should not have so few breathing apparatus that there will be danger of losing men. If we have only two sets of apparatus and some emergency occurs, the men who are near go in, and there would be hardly any hope of their being rescued if themselves endangered. There would be the danger of inducing these men to do something they should not undertake, and personally I think we ought to pass a resolution calling for not less than five helmets to a station or none at all.

J. C. ROBERTS. I believe Mr. Reese's suggestion if carried out would be a positive menace. Every life lost when helmets are used has been because there were only two men equipped with them, and there have been a number of such instances, although I know of one case where two men so equipped went in and saved a man's life. I think it is better to have no helmets at all than only two or three, and that the whole force should be combined at one station.

G. H. HAWES. Our company has seven stations with three helmets at each, and we feel that we can respond with three helmets at any time and can procure another three helmets within two hours. I believe with Mr. Reese that a limitation of five at each station is unnecessary, as we could have three helmets on the scene at the time of the accident and three more within two hours, and possibly more.

J. W. PAUL. I have outlined an amendment that might possibly cover the question raised by Mr. Reese: "Such equipment should be so located as to admit of its assembly in one hour at a central point for emergency use."

J. P. REESE. I would accept that as covering the point. I wish to point out in defense of our position that the rescue workers don't have to bother with cost sheets, and some of us managers do; and there is something besides rescuing life that we do with these breathing apparatus. For instance, we expect to save the price of our equipment by putting out coal fires with them and thus prevent the sealing of a part of the mine, as has been customary for years. Now when our men find a coal fire and it is too far for them to take the chemical extinguisher without breathing apparatus, if we keep two or three helmets at each mine we think the men can don the helmets, take in the chemical tank, and put out the fire. We do not want to go against the regulations of this conference in order to provide for that contingency, which in dollars and cents would amount to a great deal in the course of a year, but the amendment by Mr. Paul would permit us to be in harmony with this conference and yet keep only two helmets at each mine. I think we have nine helmets for our three mines, and we would not want to have five at each mine

when we can get all nine assembled in half an hour if the roads permit of automobile travel, and in a hour if a team of mules must be used.

J. C. ROBERTS. Mr. Reese was speaking of mine fires. I happened to be present at one mine where if the company had had more than two helmets it could have put the fire out. The men went down to the fire with two helmets, the oxygen in the cylinders gave out, they had to go to the surface for more oxygen, so that before they got back the fire was beyond their control and the mine had to be flooded. That is one of the bad features about two helmets. I did not arrive at the mine until after the lower workings had been flooded. I know of a number of cases where after a fire occurred the apparatus was carried to the fire in an automobile and in an hour's time the fire was out, and the apparatus more than repaid the company for the money expended for it.

AUSTIN KING. In the Connellsville, Pa., field we have large companies that have 30 to 40 collieries, and a resolution of this kind would probably embarrass them. We don't want to lose sight of the fact that the resolutions made here will be taken as a guide to work by, and any company that does not follow them would be open to censure. The practice of two of the large companies in our field is to have two apparatus at each mine, and the president and the vice president of the company told me that the purpose of those apparatus is to guarantee a sense of security to miners just as the safety lamp does at the present time. As Mr. Reese said, incipient fires can be put out if such apparatus is available, but nearly all our collieries are so situated that they are within 30 minutes' ride by wagon, and in the case of a recent fire the apparatus was brought 13 miles in 20 minutes; we had eight apparatus on the ground to reenforce two that were there from the first. So I think Mr. Paul's amendment is a good one.

I therefore move the adoption of the amendment of Mr. Paul. Anyone familiar with the regulations of Great Britain, Belgium, and, I believe, France, knows that they require but two helmets, not for the purpose of having a station, but just for the purposes that Mr. Reese mentioned.

The amendment was adopted.

The secretary read resolution 4, "The keeping of birds and mice at rescue stations for the purpose of detecting noxious gases is desirable."

AUSTIN KING. I would amend that resolution to read "carbon-monoxide gas" instead of "noxious gases," as that is the only gas for which birds are used to detect.

The motion to amend was put and adopted.

G. H. HAWES. Wouldn't carbon dioxide also be a dangerous gas that could be detected by the use of birds?

The CHAIRMAN. It is the opinion of the chemists of the bureau that it is impractical to test for carbon-dioxide gas by the use of birds and mice.

J. P. REESE. What purpose do birds and mice serve in ordinary rescue work?

The CHAIRMAN. The purpose of the use of birds and mice in rescue work is to detect the presence of carbon-monoxide gas in small percentages. They are more easily affected than men, and the fact that they topple over much more quickly than a man in an atmosphere that would be deadly to man if breathed for a long period of time makes them most valuable as a guide to the presence of this dangerous gas. As soon as a bird topples over the rescue men know that it is dangerous to proceed beyond this point without the use of breathing apparatus. By the use of birds in this way we believe we have prevented many

serious accidents from men rushing into deadly percentages of carbon monoxide unknown to themselves.

J. J. RUTLEDGE. You will find a great many men past middle age who seriously object to wearing the breathing apparatus. They do not object, however, to taking a bird with them into a questionable atmosphere, and since these men are very valuable assistants in the way of erecting brattices and restoring ventilation following a disaster, you can understand that the bird is in such instances a valuable safeguard to their lives. I believe every one who has made use of the birds at recent disasters will agree with me that they have taken a very important place in the saving of the lives of the rescuers.

G. S. RICE. The bird is of great value in closing off fire areas. At such times you all know from experience it is always found necessary for men to do work in bad air. By the use of a bird a man with a breathing apparatus can go in to the point where the work must be done, test the atmosphere with the bird, and determine its safety or danger. Besides, the bird can be retained by the men so doing the work, and any change of the atmosphere may be quickly noted by the effects on the bird.

The secretary announced that question No. 2 and question No. 4 of Section F of the preliminary program had been referred to the general conference.

Dr. A. F. KNOEFEL. In regard to the program subjects 2 and 4, "Physiological effect of pressure on the head due to wearing helmet; physiological effect of pressure of nose clip," and the "Diet of men doing training work," we might talk all morning and arrive at no conclusion. In a matter of this kind only definite information based upon careful experiment is of any value. Therefore, I move you that the Chair appoint a committee of three doctors and three rescue men to report at the next meeting of this conference upon the physiological effect of the pressure of the helmet and the proper diet for mine rescue men.

The motion was seconded and carried unanimously.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 2, RESCUE OPERATIONS.

The CHAIRMAN. Before proceeding with the reading of the startling array of details contained in the resolutions of this committee, I call your attention to the fact that the members of this committee are men of wide experience in mining operations, and they have undoubtedly adopted only such resolutions as their practical experience would seem to commend.

G. S. RICE. I believe it was Napoleon who said, "An army travels on its stomach," and since resolution 9,[a] as offered by the committee, is so complete I would suggest that it be made to cover the provision of a satisfactory commissary at mines where disasters have occurred. I myself have often seen ill effects of the failure to provide proper food for those who are engaged in rescue work.

I have in mind a recent disaster where no provision was made for food and shelter for the men, and besides the overwhelming conditions they confronted in the mine they were more seriously handicapped by this item of subsistence. After some time the Red Cross appeared on the scene and furnished relief in this matter, and later it was strengthened by some local organizations. I believe that this matter is so important that it should receive special consideration, and one of the things which occurs to my mind is the satisfactory

[a] See p. 13.

transportation of foodstuffs from the outside to those who are working in the mine.

ATHERTON BOWEN. I believe that the committee's action on resolution 9 is very good as far as it goes, but I have read Mr. Garforth's "Recovery of Mines after Fires and Explosions," and many of the things of value contained therein are omitted in this report of the committee. I should think a report of this kind should receive more deliberation and that Garforth as well as other literature on the matter should be consulted before such a report is made.

AUSTIN KING. I have read Mr. Garforth's work and from my experience many of his rules do not apply to American conditions. I call especial attention to article 2 of the ninth resolution, which reads as follows: "There should be a man in charge of outside arrangements who should see that the fan is put in readiness for operation when required." I suggest that instead of the words "the fan is" we change to "ventilation appliances are." In regard to Mr. Rice's suggestion, I suggest that section 6, which is as follows: "Proper provision to be made to take care of parties engaged in rescue work," be made to read. "Proper food and shelter should be provided for parties engaged in rescue work." Under "Inside Organization" I would reverse the order of the last two squads, making the material squad follow immediately after the temporary-ventilation squad, which it seems to me is the practical order. As reported by the committee this portion of the resolution reads, "The squads are to advance in the following order: (*a*) Helmet or advance squad, (*b*) stretcher squads, (*c*) temporary ventilation squad, (*d*) more permanent ventilation squad, (*e*) material squad."

ATHERTON BOWEN. I did not mean to say that Mr. Garforth's book was to be accepted as a whole, but it does seem to me that we are laying down a set of rules here with too little deliberation—only two days' time.

The CHAIRMAN. I assume that practically all of the members of the committee are familiar with Mr. Garforth's book and that after a sifting process have recommended only those things which they think are of importance to our conditions.

J. P. REESE. The committee did not consider that they were writing the last word on this subject; they simply attempted to get a starting point. Everything we do here is embryonic. We realize that if we go into details we could create an encyclopedia on this subject and then in the event of a disaster occurring we wouldn't know where to find the book of rules. The only idea of these few simple rules is to create an interest among mining officials and mine workers in this work. If these resolutions are read by a man he will certainly remember that in the event of a disaster the first thing to do is to organize his top work and protect his openings and be ready for the prosecution of the inside recovery work as soon as volunteers or his neighbors arrive.

I sympathize with Mr. Rice in the position he has taken in regard to subsistence. The best we have usually around a mining camp is a fourth-rate boarding house, but to one who has read these rules it would certainly occur to his mind that he would advise this boarding house to lay in a list of supplies to take care of the emergency. These rules are intended to be only a starting point and it is undoubtedly true that within the next year investigation will serve to prove that we are wrong in many of the things that we have already suggested. It seems to me that there is a danger in suggesting too much and that if we publish only a few simple rules that are proven by practice this work will do its best service to the mining public. I now move you that the three corrections suggested by Mr. King be incorporated in the report on resolution 9.

The motion was seconded and carried unanimously.

J. W. PAUL. In section *a* of the second article under "Inside Organization," which reads "Helmet or advance squad," I would suggest that "breathing apparatus" be substituted for the word "helmet," which is more or less misleading.

CHARLES ENZIAN. I have noticed frequently in these proceedings that the word "helmet" is used when "breathing apparatus" is intended and in order to avoid confusion I move you that "breathing apparatus" be accepted by this conference as the proper designation for mine-rescue apparatus. In this way we shall avoid any confusion and the word "helmet" may be retained as it was originally intended, as that part of the apparatus which is worn on the head.

The amendment was carried unanimously and the secretary was instructed to make the necessary change in the resolution.

Resolution 10 was read, as follows:

"The maximum distance rescue crews should proceed beyond fresh air: We recommend that owing to the different conditions in different mines and the hazardous work undertaken, this question should be left to the decision of the captain in charge, in conjunction with the mine officials and the probability of being able to save human life, using the time limit on all explorations."

AUSTIN KING. I suggest that in resolution 10 the word "captain," which is not generally understood, be replaced by the word "official," and this be made to read "official in immediate charge."

The secretary was directed to make the change as suggested.

Dr. A. F. KNOEFEL. I move you that consideration of the program subjects G, 5 and 6, "Cumulative Effect of Inhaling Poisonous Gases" and "Use of Stimulants," which have not been reported on by this committee, be referred to the executive committee for action.

The motion was adopted unanimously.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 3, SAFETY LAMPS AND ELECTRIC LAMPS.

The secretary read resolution 13, as follows: "Only such safety lamps and electric lamps as have passed the tests of the United States Bureau of Mines should be used."

J. W. PAUL. I would suggest the insertion of the word "types," so as to make this resolution read "only such types of safety lamps."

The secretary was ordered to make such change.

G. H. HAWES. I suggest that this resolution does not necessarily apply to metal mines, where we are not troubled with explosive gases.

AUSTIN KING. Mr. Hawes should read of the experience of Sir Le Neve Foster and Dr. Haldane in regard to some fires in English metal mines. As I remember it, a large part of these reports was made up of their investigations in regard to the carbon monoxide present.

G. H. HAWES. Weren't those mines sulphide mines?

AUSTIN KING. I do not remember as to the class of ore, but the report I speak of I know concerned largely the tin mines of Cornwall and the lead mines at Sneafell, Isle of Man.

G. S. RICE. The only trouble we have had in our western mines is from the fires in the sulphide ores, which of course do not create an explosive gas. So far as I know, no explosions have ever occurred from the products of a timber fire.

The CHAIRMAN. The question involved is that of safety lamps, and it seems to me that, as Mr. Hawes suggests, in metal mines this resolution should not prevail.

AUSTIN KING. It might meet the objection to strike out the word "and" after "safety lamps" and substitute the word "or."

J. W. PAUL. This is being considered under the heading of "Use of lamps in mine rescue and recovery work." To provide for the metal mines it could be amended by starting out with the insertion of the words "in coal mines."

AUSTIN KING. In view of Mr. Paul's suggested change I withdraw my objection in regard to the substitution of the word "or" after "safety lamps." I also second Mr. Paul's motion in regard to the change in this resolution.

The motion was passed unanimously.

G. H. HAWES. I should like to know whether sulphur dioxide in certain quantities is explosive.

G. S. RICE. It is my opinion that sulphur dioxide is not explosive, but that a certain quantity of hydrogen sulphide might be formed from sulphide ores which would be explosive under certain conditions. I suggest that our chemist, G. A. Burrell, be asked as to the explosibility of the gases in question.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 4, FIRST-AID METHODS.

The secretary read resolution 18, as follows:

"Be it further resolved that it is the concensus of opinion of this committee that a man injured with a broken back should be handled with as little movement as possible. If found in any other than a recumbent position he should be kept in that position; if found in a recumbent position apply posterior splints extending from head to feet or lay upon rolled blankets."

After some discussion by the doctors it was agreed that the rolled blankets referred to in this resolution were to be laid parallel to the spinal column to prevent the contact of the spinal column with any external material in case the patient was found in a recumbent position; in case the patient was found in any other than a recumbent position the use of rolled blankets was not to apply. In case splints were not at hand the blankets were to be rolled upon some such object as a drill bit, the patient to be laid thereon, placed on the stretcher, and firmly bound to the stretcher.

Dr. A. F. KNOEFEL. I move that the latter part of resolution 18 be changed to read: "A patient with a broken back should be handled with as little movement as possible. If found in any other than a recumbent position he should be kept in that position. If found in a recumbent position, posterior splints, well padded, extending from head to feet, should be applied, or the patient should be laid upon rolled blankets extending along either side of the spinal column, and the patient should be firmly fixed to the stretcher.

The amendment was carried.

The secretary read resolution 19, as follows:

"Be it further resolved that it is the consensus of opinion of this committee that in the treatment of all fractures of all long bones it is necessary to apply splints long enough to fix the joint above and below the fracture. For example, if there is a fracture of the tibia you have to apply the splints so that they extend below the ankle and above the knee. Be it further resolved that long splints are to be recommended in the fractures of the lower limbs."

Dr. R. F. McHENRY. I suggest that the word "tibia" be replaced by the word "leg."

Dr. W. S. ROUNTREE. We came to the conclusion at our committee meeting that a first-aid man ought to know what the tibia is, and if he does not he is perhaps incapable of doing first-aid work.

Dr. A. F. KNOEFEL. A certain education in this work is needed, and a man who does not know the forearm from the arm or the leg from the thigh is, in my opinion, absolutely incompetent for first-aid work; but I am willing to accept Dr. McHenry's suggestion to call it a leg.

There being no objection, the secretary was ordered to make this correction.

The secretary read resolution 21, as follows: "The triangular bandage should be used in preference to the roller bandage."

Dr. W. S. ROUNTREE. We have always taught the use of both the triangular and the roller bandage. We often find a case in the mine where roller bandages are at hand but there are no triangulars. In such an event a first-aid man trained in the use of the triangular only is at a loss to know how to apply the proper dressing.

Dr. A. F. KNOEFEL. The idea in adopting this resolution was that it would be a guide in our work in contests. It does not mean that a first-aid man should not be taught the correct use of the roller bandage, but this instruction should not be given until after he has shown sufficient skill in the use of the elementary triangular bandage. For the first course in bandaging, I believe that the triangular is to be preferred.

The CHAIRMAN. We should bear in mind that we are considering the matter of first-aid methods and not of first-aid contests.

Dr. M. J. SHIELDS. In making the rules which have been adopted rather generally for contests, there is one which reads like this: "Equal credit will be given for the correct use of either the roller or the triangular bandage." This rule has been very generally accepted.

Dr. W. S. ROUNTREE. If this resolution is not to affect first-aid contests I heartily agree with Dr. Knoefel that the triangular bandage is much more easily applied by those who know but little of first aid.

Dr. R. T. McHENRY. We should keep in mind the man who is injured in the mine. The preference for the triangular bandage is based on its simplicity and efficiency, and if we do not believe that this bandage should be given preference on account of its simplicity and efficiency for the immediate help of a man who is injured, we have no right to recommend it.

The CHAIRMAN. If no amendments are offered to this resolution I shall instruct the secretary to read them as read in the permanent report of the committee.

The secretary read resolution 26, as follows:

"Be it further resolved that it is the opinion of this committee that in moving an injured man he should be handled by the first-aid corps on the same side as his injury; in other words, the injured side should be next to the man lifting the injured patient. Be it further resolved that the injured man should have the right of way from the place where he received the injury to the top of the shaft, or the mouth of the slope, in all cases."

W. D. ROBERTS. Some mines do not have a shaft or a slope; they are drift mines; and I suggest that this be made to read "to the surface" instead of "to the shaft or mouth of the slope."

There being no objection, the secretary was directed to make the change.

H. R. OWENS. In the event of injury to several men it might be necessary to bring rescue appliances and rescue men from the surface, and in this event I do not believe it would do any harm to allow the injured to wait until such reenforcements arrive.

AUSTIN KING. I believe that "a bird in the hand is worth two in the bush," and that we ought to take care of the one we have and give him the right of way.

Resolution 26 was passed without further objection.

The secretary read resolution 27:

"Be it further resolved that this committee would recommend, if this organization is perfected, that a committee of seven persons—two first-aid men, two operators, two physicians, and one representative of the Bureau of Mines—be appointed to act as an advisory board or as an executive board. This committee will have the power to accept or reject any new dressings that may be offered in any field contest."

Dr. A. F. KNOEFEL. I was chairman of this committee, and resolution 27 is to be effective only in the event of the establishment of a permanent organization. The allotment of this committee among men with special training in various lines of mining may be changed if our allotment does not meet the approval of this conference. It will be the duty of this committee to accept or reject any new ideas or dressings that may originate in any field contest or in the ordinary course of training for first aid.

The CHAIRMAN. Would it be agreeable to the chairman and to the members of committee 4 if the resolution (No. 27) be changed so that it may be referred to the executive committee of the permanent organization?

Dr. A. F. KNOEFEL. I scarcely believe it policy to burden the executive committee with the work designated in resolution 27. The executive committee will enter upon its duties immediately, but the committee referred to in resolution 27 should be appointed only after the permanent organization has been established.

J. W. PAUL. I make a motion that resolution 27 be referred to the committee on permanent organization for its consideration.

The motion was seconded by C. G. Brahm and carried.

Dr. R. F. McHENRY. I call attention again to resolution 20, which reads as follows: "A first-aid man should not be allowed to reduce a dislocation." I want to amend that to read, "except dislocations of the lower jaw or a finger."

Dr. M. J. SHIELDS. I would suggest a change in the text of the resolution as it stands, in that it should read, "It should not be the duty of a first-aid man to reduce a dislocation."

Dr. W. S. ROUNTREE. I still believe that the motion as it now stands should prevail. In this day of enlightenment you can get a physician in most any mining camp in from 15 minutes to an hour's time. The first-aid man does not know how to diagnose the symptoms of a dislocation, and it is my experience that he will oftentimes mutilate the ligaments and leave the injury in far worse condition than it would have been if he had done nothing at all with it.

T. B. DILTS. I think with Dr. Shields that the words, "It should not be the duty of" would cover that, instead of "A first-aid man should not be allowed to." Then in the event of a first-aid man being competent to reduce a dislocation he would feel free to do so, and thus relieve the suffering at once. If, on the other hand, he should feel incompetent through lack of experience he would fix the joint in position and allow the dislocation to be reduced by the physician.

Dr. R. F. McHENRY. I move you that the resolution be made to read, "It should not be the duty of the first-aid man to reduce a dislocation except of the lower jaw or a finger."

The motion was seconded by Mr. Reese.

The CHAIRMAN. With the permission of Dr. McHenry I shall first put a motion to amend the resolution as it stands to read, "It should not be the duty of a first-aid man to reduce a dislocation," instead of "A first-aid man should not be allowed to reduce a dislocation."

The motion was carried unanimously.

The CHAIRMAN. I shall now put a motion to amend the resolution still further to read, "It should not be the duty of a first-aid man to reduce a dislocation except of the lower jaw or a finger."

The motion was put, with the result that 16 voted in the affirmative, and 16 in the negative.

Dr. A. F. KNOEFEL. If it meets with Dr. McHenry's approval, I should like to add the word "simple," making his amendment read, "Except in the case of a simple dislocation of the lower jaw or of a finger." It is easy for first-aid men to identify a simple or a compound dislocation, and it is my opinion that he should reduce the simple dislocation and do nothing with the compound dislocation except to apply the compress dressing.

Dr. W. S. ROUNTREE. I still insist that the ordinary first-aid man is incompetent to judge the difference between a compound and a simple dislocation. Consequently it is better to advise that it is not his duty to reduce any dislocation; then in the event of a simple dislocation there will not be much cumulative effect due to its nontreatment, and in the event of a compound dislocation it will be very much better and more easily treated when the patient gets in a doctor's hands.

Dr. R. F. McHENRY. The diagnosis of a dislocation is very simple indeed. Most any first-aid man knows that in a compound fracture you have a hole extending from the outside down to the fractured bone, and that in a simple

fracture you do not. Probably no dislocation is so easy to identify as a dislocation of the lower jaw, and none is so easily reduced if it is attended to promptly. There is, however, no dislocation so painful as that of the lower jaw if it is not treated at once; and because of the simplicity of diagnosis and of treatment, I believe it should be within the province of the first-aid man to reduce a simple dislocation.

The CHAIRMAN. Will some of the first-aid miners debate the motion?

C. O. ROBERTS. I was down in West Virginia last winter in some isolated mining camps, places at which the services of the physician could not be obtained quicker than from one to three hours, and in conversation with mining surgeons I found that they all agreed that the reduction of a simple fracture of the lower jaw is a simple matter, and that the first-aid man should be taught to reduce it.

W. D. ROBERTS. I am very much in favor of permitting those first-aid men who have superior intelligence to reduce a simple dislocation of the lower jaw or of a finger. Dr. Rountree has, perhaps, had experience in teaching the work of first aid to colored and to foreign laborers, and with that type of men I believe the teaching should advise against the treatment.

The amendment was passed by a vote of 19 to 13.

It was then moved and voted unanimously that the resolutions of the committee as amended be adopted.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 5, FIRST-AID TRAINING.

CLARENCE HALL. Before we proceed further I should like to report from G. A. Burrell, the gas chemist of the Bureau of Mines, as to the explosibility of gases given off from sulphide ores. Sulphur dioxide (SO_2) is of course nonexplosive, but a certain amount of hydrogen sulphide (H_2S) may be generated from sulphide ores, and this is an explosive gas, forming H_2O and SO_2 upon combustion. However, it is usually found in a small quantity only and the amendment of resolution 13 as to the electric and safety lamps for use in metal mines should stand as it is at present.

J. J. RUTLEDGE. I suggest that in adopting the resolution which Mr. Hall mentions, as it stands, we are excepting one class of mining where all the precautions necessary in coal mining must be taken—that is, asphalt mining, where inflammable gas and dust occur and where already one great disaster has happened.

The chairman announced that without objection resolution 13 would stand adopted without further amendment.

The secretary read resolution 30 as follows:

"Successful first-aid work at mines must have the personal interest of the company officials, the financial support of the mining company, and the cooperation of the mine physician, surgeons, and employees."

AUSTIN KING. I should like to ask what is intended by "financial support of the mining company."

Dr. A. F. KNOEFEL. There are various little expenses connected with first-aid work, such as the purchase of the necessary supplies to be used in training, the purchase of first-aid materials for the first-aid stations underground, the payment of expenses of the first-aid team to and from the points of the contests.

This is the financial support which the results of first-aid work justify the company in giving, and without this financial support first-aid work will never be a success at any mining center.

The secretary was directed to read resolution 31, as follows:

"Every mine should have a sufficient number of first-aid men on duty to take care of any injured persons throughout the 24 hours of the day."

AUSTIN KING. I believe that some modification of the phraseology of the resolution should be made to suit the condition where only a very few men are employed on the night shift. In the event of large repair gangs being employed at night the instructions of this resolution would be practical, but where only two or three men are employed underground at night I doubt the practicability of the resolution.

Dr. A. F. KNOEFEL. The idea of this resolution was not to fix any definite number of first-aid men required to a certain definite number of men. The intention was that each operator should judge his own conditions and make his own decision as to the number of first-aid men he would require. We have so framed this resolution that it is liberal in this respect. If we should have named a definite number of first-aid men as being desirable among each 100 employees, for instance, a mining company would be open to censure if, in the event of a disaster, it could be determined that such a number of first-aid men had not been provided. We already have too many of that undesirable class, known as "ambulance chasers," and it is our desire to give them as little food for scandal as is possible.

The resolutions presented by committee were passed unanimously as amended.

The SECRETARY. At the bottom of the report of committee 5 I find a notation to the effect that the report on hospitals, which was to be made by committee 7, is the result of joint action by committees 5 and 7, and since that result is attached hereto I believe it would be well to consider the resolution which pertains to hospitals at this time.

The secretary then read resolution 36, as follows:

"Underground surgical hospitals are unnecessary, but there should be deposited at different points in the mine a sufficient number of first-aid packets properly equipped with first-aid emergency dressing. In addition there should be located a sufficient number of first-aid dressing stations at the bottom and immediate surface opening of the mine."

AUSTIN KING. I think the word "and" in the last sentence should be made "or."

Dr. A. F. KNOEFEL. As chairman of this committee I wish to say that we anticipated the suggestion of this change, but we do not agree to it. Many men are injured on the top, and they require immediate attention, and we should not be compelled to bring first-aid materials from the mine and interrupt operations in order to give such men proper attention. Also many men are injured in the immediate vicinity of the shaft bottom, which is generally accepted as one of the busiest points in a mine. It is very necessary to have supplies immediately at hand to take care of the accidents that are bound to occur at this point.

AUSTIN KING. The difference between the bottom of the shaft and the surface is only one-half minute. I am not pleading for my own company, for in most instances we have complete hospital rooms, steam heated and supplied with hot and cold water, both at the bottom and at the top; but I do believe that it is an injustice to ask a small operation to prepare suitable conveniences at both the top and the bottom where there is only a half minute's time between the two.

Dr. W. S. ROUNTREE. I heartily agree with Mr. King. Ordinarily, if a man is injured in our mines first-aid materials are brought to the scene of the injury and he is skillfully treated by the first-aid man, placed on an ambulance car, which is promptly taken to the surface, and in the dressing room on the surface he awaits the arrival of the ambulance.

G. S. RICE. I believe it is a good plan to have a small station at the bottom and a more complete station on the top. It is not always possible to get the cage quickly when you want it, and the injured man is compelled to wait in a strong draft of cold air. A very happy arrangement is to combine the underground office and the hospital in one room.

J. P. REESE. I have found the underground hospital at the shaft bottom to be very important. In the case of many men injured below, when a quantity of supplies is needed on any particular entry it forms a base of supplies from which materials can be quickly dispatched with the minimum amount of confusion.

Dr. A. F. KNOEFEL. The resolution was framed so as to be lenient with the mining company in the way of expense. You note that we do not recommend extensive underground hospitals, but all that we do ask is a room at the bottom where the man may be placed upon a cot, removed from the excitement of mules, motors, and men.

W. D. ROBERTS. I remember where several men were injured underground when the cage rope broke. The first-aid men descended by means of ladders and took care of the injured men, but there was no available underground station and the injured men suffered greatly because they could not be hoisted for several hours.

JESSE HENSON. The anthracite law requires that there be a hospital situated underground and another at the surface.

AUSTIN KING. What we are trying to recommend is a resolution adapted to the great bulk of operations.

Dr. R. F. MCHENRY. It has been reported to this committee that there is in this immediate district a very complete underground hospital, equipped with everything but surgical instruments. Now, this is the thing we want to condemn, for it is, perhaps, an unnecessary expense; but what we do ask for is not merely a hole in the wall but a room properly lighted and heated, with hot and cold water—a brick or cement-lined first-aid station. This can be constructed at a very reasonable expense, and its value is equal to that of the elaborately equipped underground hospital.

CHARLES ENZIAN. I move that the resolution of Mr. King be amended to read that "if practicable" first-aid stations should be installed at the bottom of the mines.

The motion was not seconded.

J. W. PAUL. I move that the word "surgical" be inserted previous to the word "hospital."

Mr. Paul's motion was put and carried.

The original resolution as amended by Mr. Paul was passed.

W. D. ROBERTS. I would like to amend the resolution to read, "First-aid rooms properly equipped should be placed at the bottom and at the top."

"Equipment" would include blankets and stretchers, and it would not be necessary to have stretchers all over the plant.

The CHAIRMAN. It is the practice of some companies to deposit at first-aid stations—not dressing rooms, but mere stations, far away from the bottom of the shaft—first-aid boxes or canisters containing blankets and other first-aid necessities.

J. W. PAUL. These are recommendations made by this conference. If any coal company wishes to act upon these recommendations it will confer with some person conversant with the equipment of these stations and will properly provide them with the appliances and the dressings needed. If the legislatures ever enact laws specifying what these stations should contain I think that those who pass on these suggestions will see that they are properly equipped.

The CHAIRMAN. The motion stands as passed, without further modification.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 6, FIRST-AID CONTESTS AND EXHIBITIONS.

The SECRETARY. With the permission of the chairman I shall ask J. C. Roberts, secretary of committee 6, to read its resolutions, as they are in his own handwriting.

Mr. Roberts read resolution 34, as follows:

"The judges should be of sufficient number so that one judge should handle not more than three teams, preferably less, and these judges should rotate. These judges should be first-aid surgeons and practical first-aid men (not surgeons). Each contest should consist of not more than five events."

Mr. Roberts proceeded to explain that the words "first-aid surgeons" were included for the reason that not all surgeons are familiar with first-aid methods, a fact that was brought to his attention in the West.

CHARLES ENZIAN. There has been a suggestion of an element of professionalism in the anthracite district of Pennsylvania in these contests, and an attempt is being made to eliminate this by substituting the name "exhibition." It has been experienced that when a man is specially capable in first-aid work the captains of other first-aid teams will go far in their efforts to procure him for their teams. I believe contests should be limited to interstate affairs and exhibitions should be held as intermine or intercolliery affairs. A number of companies in the anthracite region have adopted this distinction.

The CHAIRMAN. The condition described by Mr. Enzian has been recognized by certain of the big companies who now oppose contests, and the experience of the anthracite-mine people in this respect may be your experience very shortly.

CHARLES ENZIAN. I am in favor of making these affairs an exhibition of skill rather than a contest. The chairman of our companies now invite their salaried men to attend these exhibitions at the company's expense, to note what is being done by the company for the purpose of insuring greater safety to life. This develops a valuable esprit de corps.

RESOLUTIONS OF COMMITTEE 7, HOSPITALS AND SAFETY STATIONS.

Committee 7 presented no resolutions, as its business had been transacted in the joint sectional meeting of committees 5, 6, and 7.

DISCUSSION ON RESOLUTIONS OFFERED BY COMMITTEE 8, PERMANENT ORGANIZATION.

Resolution 37 was read by the secretary as follows:

"*Resolved*, That this conference be made a permanent organization; that the presiding chairman of this conference, H. M. Wilson, be elected the president of a temporary committee to be composed of seven persons; that there be elected a vice president; that C. S. Stevenson be elected secretary, and that the other members of the committee be appointed by the president; that the purpose of the temporary committee shall be to draw up a constitution and by-laws and otherwise complete the organization; and that until there be such permanent organization all details in connection with first aid and other matters be passed over."

Dr. A. F. KNOEFEL. In explanation of the last clause of resolution 37 I wish to explain that it was the sense of the committee that it should simply lay the groundwork, and that the details, such as first-aid dressings and appliances, should be left to the permanent organization.

After some desultory debate the committee's resolution was rephrased and adopted.

The CHAIRMAN. I shall now appoint the temporary executive committee provided for in the last resolution. This committee is apportioned in accordance with geographical position and also among mine officials and doctors, as follows: Drs. F. L. McKee for the East, W. S. Rountree for the South, and A. F. Knoefel for the Middle West, mine official E. H. Weitzel for the Far West, Austin King for the bituminous region of this part of the country, John P. Reese for the central and western coal fields, G. H. Hawes for the northern iron and copper regions, and R. A. Phillips for the anthracite coal fields of eastern Pennsylvania.

The preliminary meeting of the executive committee will be convened at 2 p m. to-day.

Before a motion to adjourn is entertained I wish to express to you the sincere appreciation of the Director of the Bureau of Mines and of the employees of the bureau for the manner in which you have come from all parts of the United States and remained with the conference during three long days. We did not know in advance whether or not much would be accomplished, but I think everyone must admit that this is a most representative gathering, even if it is not very large numerically. You have done splendid work, and a great deal of it, in a short time, and I think that later you will all be proud of your part in this preliminary meeting, which I believe will mean a great deal for the mining people of the United States.

In the past we had gone through the period of litigation following accidents, followed for some years by employers' liability legislation, but in the last year a tremendous impetus has been given to the enactment of legislation—Federal and State—providing for workingmen's compensation. In the year 1911 ten States enacted laws providing compensation for injuries received by workmen. It will therefore be wise for the operators, in the interest of economy as well as for humanitarian reasons, to adopt the slogan originated at the first national mine-safety demonstration a year ago—"Safety First." I think this conference is the beginning of a big movement. I thank you heartily for your attendance.

Dr. M. J. SHIELDS. Before we adjourn I should like to have permission to express the thanks and appreciation of the Red Cross for the work of the representatives of this conference.

Dr. A. F. KNOEFEL. I second the sentiment of Dr. Shields and especially mention Mr. Wilson, who has been to us like a kind father with a bunch of unruly children, and I think that in support of this resolution we ought to have a rising vote of thanks.

J. P. REESE. Before the vote of thanks—I think we will all vote for it very willingly—according to the decision regarding permanent organization this body should appoint a vice president.

On motion duly seconded J. P. Reese was unanimously elected vice president.

J. P. REESE. I wish to thank you for the unexpected and unsolicited honor, but I earnestly hope that the president will continue in good health. I now put the motion of the doctor to thank the members of this conference, and especially our untiring chairman.

The motion was passed unanimously by a standing vote.

On motion, the conference adjourned at 1.30 p. m., with the understanding that the temporary executive committee would meet immediately after lunch.

PROCEEDINGS OF THE TEMPORARY EXECUTIVE COMMITTEE.

The members present were: H. M. Wilson, president; J. P. Reese, vice president; C. S. Stevenson, secretary; Dr. W. S. Rountree, Dr. F. L. McKee, G. H. Hawes.

The chairman outlined the duties of the committee, laying emphasis on the duty of adopting a constitution and by-laws, and naming the time of meetings.

Dr. A. F. Knoefel suggested that a subcommittee be appointed by the chair to draw up a skeleton constitution and by-laws, copies to be mailed to all members of the committee for correction; that upon receipt of the corrected copies the subcommittee should complete its work, and mail to all members copies of the revised constitution and by-laws for final correction.

G. H. Hawes moved that the chairman appoint a committee of three to act as suggested by Dr. Knoefel, and that the chairman be one of this subcommittee. The motion was carried unanimously.

The chairman appointed Dr. F. L. McKee and Austin King as a subcommittee to act with himself in preparing a constitution and by-laws.

The chairman announced the appointment of G. H. Hawes as a member of the temporary committee on permanent organization to replace J. P. Reese, who had been elected vice president.

Dr. W. S. Rountree moved that the conference meet annually, and that the executive committee announce the date of these meetings.

The motion was carried unanimously.

Austin King stated that the purpose of the organization should be to promote safety in mines by means of the adoption of first-aid measures and logical methods of procedure in rescue and recovery work.

The chairman, after discussion, advised all members of the committee to suggest a name for the organization within 30 days.

A discussion that remained unsettled at adjournment was as to what should constitute membership in the organization. Two suggestions were offered, as follows:

(1) Members should consist of those persons who by occupation or interest are qualified to consider the issues for which the association is organized.

(2) Each company might become a member and have one associate nonvoting member for each 2,000 men or fraction thereof employed, the membership to be extended to those organizations having allied interests.

After a discussion of the question of dues, on which no definite action was taken, the meeting adjourned.

PROPOSED CONSTITUTION FOR A PERMANENT NATIONAL MINE-RESCUE AND FIRST-AID ASSOCIATION.

After the adjournment of the conference the temporary executive committee adopted the following constitution, which will be recommended to the next meeting of the conference, to be held in September, 1913, in accordance with article 3 of the constitution. The draft is presented here for the information of persons concerned in the safety of mining operations.

ARTICLE I.

NAME.

SECTION 1. This organization shall be known as the American Mine-Safety Association.

ARTICLE II.

OBJECT.

SECTION 1. The object of the association shall be to promote the science of safety in mines and mining by the adoption of improved first-aid methods, and of logical methods of procedure in rescue and recovery work; to recommend the adoption of approved types of first-aid and mine rescue and recovery appliances; to obtain and circulate information on these subjects; and to obtain the cooperation of its members in establishing proper safeguards against loss of life and property by explosions or fires or from other causes.

ARTICLE III.

MEMBERSHIP AND DUES.

SECTION 1. Membership shall consist of (*a*) active, (*b*) associate, (*c*) life, and (*d*) honorary members. It is understood that membership pledges no one to any given course of action.

SEC. 2. Active members shall be persons who, by occupation, affiliation, or interest, are qualified to consider the issues and forward the aims for which the association is organized. Annual dues shall be $3.

SEC. 3. Associate members shall be national institutes, societies, and associations; National and State departments and bureaus; and firms and corporations concerned in the mining industry or whose principal object is the reduction of the loss of life and property in mines. Annual dues shall be $10, except that the associate membership is hereby granted to the Federal Bureau of Mines without payment of dues, because of the existence of a Federal statute that "prohibits the payment of membership fees or dues in societies or associations." Associate members shall be entitled to representation by three persons, each of whom shall be entitled to all of the privileges of active members.

SEC. 4. Active members may obtain a life membership by the payment of $100, thereby exempting them from further payment for life; associate members may obtain a life membership by the payment of $250, thereby exempting them from further payment for 25 years.

SEC. 5. Honorary membership may be conferred upon persons who for more than fifteen (15) years have rendered exceptional service in the work of mine safety. A favorable report by the executive committee of the association and a two-thirds majority vote by the members of the association present at an annual meeting shall be necessary to elect an honorary member. Such membership shall carry with it all the privileges of active membership, without dues.

ARTICLE IV.

OFFICERS.

SECTION 1. The officers shall be a president, a vice president, a secretary-treasurer, and an executive committee of six, of whom the president and the vice president shall act as ex-officio members.

ARTICLE V.

ELECTION OF OFFICERS.

SECTION 1. At each annual meeting the president and the vice president shall be elected for a term of one year, or until their successors are elected, and two members of the executive committee shall be elected for three years; the executive committee shall elect its own chairman immediately after the general election.

SEC. 2. The secretary-treasurer shall be appointed by the executive committee from the active membership, and shall hold office during the pleasure of said committee.

SEC. 3. Any candidate for an office shall be nominated in writing by at least 10 members, nominations to be submitted to the secretary at least four weeks before the annual meeting.

SEC. 4. The candidate for any office receiving a majority of the votes of the members present at the annual meeting shall be declared elected.

ARTICLE VI.

DUTIES OF OFFICERS.

SECTION 1. The president shall perform the duties of the presiding officer and chairman of the association. He shall also be ex officio a member of the

executive committee and of all other committees, and may appoint thereon as his proxy any other officer of the association. He shall also sign all orders for the payment of money.

SEC. 2. The vice president shall perform all the duties prescribed for the president, in his absence. The vice president shall also be ex officio a member of the executive committee.

SEC. 3. The secretary-treasurer shall prepare a program for and keep a record of each meeting, countersign all orders for money that have been signed by the president, keep a true record of all cash received and disbursed by him on account of the association, and publish such proceedings of the association as the executive committee may authorize. He shall also act as secretary of the executive committee.

SEC. 4. The secretary-treasurer shall receive a salary to be fixed by the executive committee, and his annual dues shall be remitted during office.

SEC. 5. The executive committee shall pass on new members, audit the accounts of the treasurer, and approve the programs of meetings and all papers to be read before or published by the association.

SEC. 6. At any meeting of the executive committee, five members of the committee shall constitute a quorum for the transaction of business.

ARTICLE VII.

COMMITTEES.

SECTION 1. The technical business of the association, or such as relates to the consideration of methods of first-aid or rescue-training work, or appliances used therein, shall be transacted by regular and special committees, the chairmen and members of which shall be appointed by the president, with the approval of the executive committee, at least within thirty (30) days following the holding of the annual meeting. Said committee shall meet on the day preceding the annual meeting, or at any regular meeting that may be called for the consideration of business in which such committee or committees may be concerned.

ARTICLE VIII.

ANNUAL AND SPECIAL MEETINGS.

SEC. 1. The annual meeting of the association shall be held in September of each year, the place of meeting to be determined by the executive committee not less than 60 days in advance of the meeting.

SEC. 2. Special meetings may be called by the executive committee and shall be called by them upon written application of twenty (20) active members. Notice of such meetings shall be sent to all members at least twenty (20) days in advance. The notice shall state the business to be transacted, and no other business shall be entertained.

ARTICLE IX.

LOCAL SECTIONS.

SEC. 1. When, in the judgment of the executive committee, a sufficient number of members and associates shall petition in writing, its members may form, subject to the constitution and other regulations governing this association, a local section for the purpose of more effectively carrying out the aims of this association. Each local section may adopt such by-laws as it may deem expedient.

SEC. 2. Each local section shall select some place as headquarters and a definite territory to be especially benefited by said local section and within which its members must reside. Both headquarters and territory are subject to the approval of the executive committee, and may be changed by the executive committee as occasion may require.

SEC. 3. All members of the association in good standing who reside in the territory set apart by the executive committee to be tributary to any local section shall be considered members of that local section and shall be so enrolled; and they shall be entitled to all the privileges that such local section may under the constitution provide. A member may vote or hold office in one local section only.

SEC. 4. The officers of each local section shall consist of a chairman, a secretary, and such committees as each section may find desirable. These officers shall be elected by the votes of the members of a section in the manner provided by the section by-laws. The election of any member as an officer of a local section shall not debar him from election or appointment to any other office in the association.

ARTICLE X.

BUSINESS YEAR.

SEC. 1. The calendar year shall be the business year of the association, and no member shall receive a copy of the proceedings of the association nor be permitted to vote at any election who shall be in arrears for dues for the current year on the first day of September of that year.

ARTICLE XI.

APPOINTMENT OF DELEGATES.

SEC. 1. In response to proper invitation the president may appoint a delegate or delegates to represent the association at a meeting of any organization represented in the association membership, and such delegate or delegates shall report the substance of the proceedings at the next succeeding annual meeting of the association.

ARTICLE XII.

STANDARD METHODS AND APPLIANCES.

SEC. 1. A proposed standard method of instruction in first aid to the injured or in rescue operations, or in training or instruction relative thereto; or in the holding of contests or other procedure relative thereto; or a proposed standard specification for approval or appliances or equipment for use in first-aid or rescue work must be presented to the executive committee, in writing, at least sixty (60) days prior to the holding of the annual meeting, at which time it must be referred to the appropriate committee and reported therefrom to the meeting, which may amend it by a majority vote. A two-thirds affirmative vote of those voting shall be required to refer the resolution to letter ballot of the association. A two-thirds affirmative vote of those voting in a letter ballot shall be required for the adoption of the standard resolution or specification.

ARTICLE XIII.

ORDER OF BUSINESS.

SEC. 1. At all meetings the following shall be the order of business:

1. Reading of minutes of preceding meeting.
2. Report of the executive committee.

3. Report of officers.
4. Report of auditing committee.
5. Report of other committees.
6. Election of new members.
7. Unfinished business.
8. New business.
9. Program.
10. Adjournment.

ARTICLE XIV.

AMENDMENTS.

SEC. 1. This constitution may be altered or amended at any stated meeting by a two-thirds vote of the members present, provided notice has been given at a previous annual meeting of the proposed change, alteration, or amendment.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. — pp., — pls., — figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

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Bulletin 63

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

SAMPLING COAL DELIVERIES

AND TYPES OF GOVERNMENT SPECIFICATIONS FOR THE PURCHASE OF COAL

BY

GEORGE S. POPE

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SAMPLING COAL DELIVERIES, AND TYPES OF GOVERNMENT SPECIFICATIONS FOR THE PURCHASE OF COAL.

By GEORGE S. POPE.

INTRODUCTION.

The purchase of coal by the Government under specifications depending on the heating value of the coal, its content of ash and of moisture, and other considerations, rather than upon the reputation or trade name of the coal, was based on the fuel investigations begun by the Technologic Branch of the United States Geological Survey in 1904. The plan was first adopted by the Treasury Department in 1906. Since then the plan, variously modified in form but the same in principle, has been gradually adopted by other departments until at present of the coal used by the Government, the total value of which approximates $8,000,000 annually, more than half is purchased under specifications. The Government publications dealing with the adoption by the Government of the specification plan, the number of contracts awarded on that basis, and the quality of the coal delivered under such contracts in the several fiscal years covered by the reports are given in publications[a] listed at the end of this bulletin.

Under the authority of acts of Congress making appropriations for analyzing and testing fuels belonging to or for the use of the United States Government, a laboratory is maintained at the headquarters of the Bureau of Mines, Washington, D. C., where samples representing deliveries of coal purchased under specifications for Government use are analyzed and tested.

At this laboratory more than 1,200 samples have been analyzed and tested in one month. With the complete equipment and the efficient force employed, the analytical work has been so perfected that the analyses and heating-value tests of the samples received are as accurate as may be reasonably expected in laboratory work of this nature.

One of the serious drawbacks to the general adoption of the specification method for the purchase of coal is the difficulty of obtaining at reasonable cost samples of coal that can be considered fairly representative of the commercial product delivered in wagons, railroad

[a] See Bulletins 11 and 41, Bureau of Mines.

cars, or ships. Therefore the method of taking and preparing samples for shipment to the laboratory has been given fully as much care as the making of the analyses and tests, and a general plan of collecting samples fairly representative of the delivered coal has been evolved through various modifications and improvements based on experience and increased knowledge of the physical and chemical characteristics of the various coals that are purchased by the Government. The method that is in general use by the Government is described in the following pages.

In connection with studies of the coal deposits of the country, of the best methods of preventing waste in mining, and of increasing efficiency in the utilization of coal belonging to or for the use of the Government, geologists and engineers of the United States Geological Survey and of the Bureau of Mines have visited more than 1,700 coal mines scattered through all of the coal-producing States and Territories. From each mine from 2 to 8 or more samples were taken, the number depending upon the size of the mine and its output. The analyses and descriptions of the samples collected up to July 1, 1910, are published in a Bureau of Mines bulletin, entitled "Analyses of Coals in the United States," which is now in course of publication.

A study of the analyses of these samples and of the samples taken from cars shipped from a number of the same mines shows that the mine samples are as a rule of higher grade than the average commercial shipments, particularly with respect to containing a lower percentage of ash and having a correspondingly higher heating value. This difference is due to the fact that the Government inspector, proceeding on the basis of what can be done by a good miner, usually removes more of the partings of bone, slate, and other extraneous matter from his mine samples than the average miner does in ordinary practice, the aim of the miner being to get the maximum number of tons past the tipple inspection, as his earnings are based on his output. The average miner, therefore, does not always take as much care as he might in rejecting the impurities. The difference may be caused, too, by the presence of pieces of the mine roof and floor. The Government inspector can readily exclude these from the mine sample, but they often get into the commercial coal as a result of the character of the roof and floor and of the mining methods employed. The efficiency of the tipple inspection and the means employed in rejecting impurities when the coal is loaded into the railroad car are also factors that often largely account for the difference between mine and commercial samples. Most of the samples collected from the mines by the Government inspectors show a higher moisture content than commercial samples, because of the generally moist atmosphere of the mine and the precaution taken to prevent loss of moisture in the collection, preparation, and analysis of mine samples.

When properly taken, the mine samples are of great value, as they indicate the general character of the coal and the uniformity of the coal bed and enable one to determine its probable value for any designated purpose, providing due consideration is given to the character of the partings, the roof, and the floor, and to the possibility of pieces from these being loaded with the commercial output.

The collection of mine samples by the Bureau of Mines and the Geological Survey is done in a systematic manner, according to a prearranged plan, and the same procedure is always followed where circumstances permit. A special outfit for use in collecting mine samples has been developed. The method followed and a description with illustrations of the outfit are published in Bureau of Mines Technical Paper 1, entitled "The Sampling of Coal in the Mine."

THEORY OF SAMPLING.

To determine with utmost accuracy the ash content and heating value of a quantity of delivered coal would require the burning of the entire quantity, and special apparatus arranged to measure the total heat liberated, or would require crushing the whole quantity, and reducing it by an elaborate scheme of successive crushings, mixings, and fractional selections to portions weighing approximately 1 gram, the minute quantity which the chemist requires for each determination. Either of these procedures is obviously impracticable if the coal is to be used for the production of heat and power.

The method actually employed is to select portions from all parts of a consignment or delivery of coal and to systematically reduce the gross sample, obtained by mixing these portions, to quantities that the chemist requires for making ash determinations or that can be burned conveniently in the calorimeter, an apparatus for determining the heating value. The gross sample should be so large that the chance admixture of pieces of slate, bone coal, pyrite, or other impurities in an otherwise representative sample will affect but slightly the final results. Increasing the size of the gross sample tends toward accuracy, but the possible increase is limited by the cost of collection and reduction. In reducing the gross sample by successive crushings and halvings or fractional selections, the object is to procure a small laboratory sample that, upon analysis, will give approximately the same results as the gross sample itself, or, in fact, the entire quantity of coal from which the gross sample was obtained.

THE METHOD OF SAMPLING AS A PART OF COAL SPECIFICATIONS.

Recognizing the importance of the method of sampling as being a definite commercial procedure and of having the method clearly set forth in the specifications to become a part of the contract, and recognizing also the desirability of insuring uniformity and similarity

in the specifications used by the different branches of the Federal service for the purchase of coal, representatives of the executive departments and independent establishments of the Government held a conference under the auspices of the Bureau of Mines in February, 1912, for the purpose of discussing these and other features of the specifications. At this conference committees were appointed to prepare specifications in accordance with the views of the members.

As these specifications state the method of sampling in more detail than previous Government specifications, the specifications which were generally used by the Government for the purchase of coal for the fiscal year 1912–13, with such modifications as have later appeared desirable, are given on pages 52–66, with the view of presenting herewith a specification which will very probably be typical of the general specifications which will be used for the fiscal year 1913–14. It was recognized at the conference that in general specifications, such as were recommended, certain requirements had to be of wide application, as the specifications cover such a wide variety of conditions, not only as to character and quality of coal but as to type of furnace equipment, size of deliveries, methods of delivering, etc.

PRACTICAL CONSIDERATIONS.

MOISTURE.

The specifications which were used for the purchase of coal on the heat-unit basis prior to the fiscal year 1912–13 were on the B. t. u. (British thermal unit) "as received" basis; that is, payment for delivered coal was directly affected by the moisture content of the sample received by the laboratory. This method was based on the assumption that the moisture in the samples collected at the time of weighing and delivery could be preserved with slight loss during the storing and subsequent working down of the gross sample to a quantity convenient for transmittal to the laboratory and in its later treatment in the laboratory. From experiments that have been made and from a large mass of data, it is known that the moisture content of coal does not remain constant, and that the moisture content reported by the laboratory may be as much as 5 to 10 per cent lower than that actually contained in excessively wet or high-moisture coal at the time of weighing.

In one investigation, 254 gross samples were reduced, at the delivery point, to samples weighing approximately 5 pounds each and then the 5-pound samples were divided into two equal parts ("duplicates"), which were placed in mailing cans and sent to the Bureau of Mines for analysis. The moisture contents of the "duplicates" were presumably identical, but analysis showed that the average difference of the moisture contents of the 254 pairs of "duplicates" was 0.256

per cent and the maximum difference was 3.6 per cent. These figures indicate the moisture differences that may exist even after the gross sample has been reduced to approximately 5 pounds, but they do not show the moisture lost by the gross sample while in storage and while being reduced to 5 pounds. As stated above, the total loss of moisture may be as much as 5 to 10 per cent in high-moisture or excessively wet coals.

As a sample loses moisture, its B. t. u. "as received" value correspondingly rises, with the result that the price for delivered coal determined on the "as received" value is, with rare exceptions, higher than that warranted by the quality of the coal at the time of weighing. As a general statement, payment based on the "as received" B. t. u. value will be higher than warranted, unless the sampling and laboratory work can be carried on under conditions that minimize moisture loss, as under freezing temperatures.

Recogizing the uncertainty involved in taking the moisture determination in the laboratory as representative of the moisture content of the delivered coal and the consequent possibility of payment of a higher price than is warranted, the Bureau of Mines recommended to the executive departments and independent establishments of the Federal service that the heating value in the coal specifications for the fiscal year 1912–13 be on the "dry coal" basis.

In preparing these specifications the fact was recognized that the amount of moisture contained in coal produced from day to day from the same mine, or group of mines working the same bed, is largely accidental, and is a matter over which the buyer and seller have only slight control. However, in order to place a negative value on high-moisture coals and to protect the Government against the delivery of coals containing excessive amounts of moisture, the specifications require the bidders to specify the maximum moisture content in coal offered. This value becomes the standard of the contract.

If coal of uniform B. t. u. "dry coal" value is delivered on a contract, the contractor receives the advantage on any delivery in which the moisture content approaches the maximum specified, because he is paid for the weight of water contained in the coal in excess of a normal amount, whereas if the coal is very dry, containing less than the normal amount of moisture, the purchaser receives the advantage.

For example, coal is delivered under a contract in which the standards are 14,300 B. t. u. per pound, "dry coal," and a maximum moisture content of 3.5 per cent. The heating value of a ton (2,240 pounds) of "dry coal" would be 32,032,000 B. t. u. Assume that the average moisture content of deliveries for a year is 2.5 per cent,

then for every 2,240 pounds of "dry coal" having a heating value of 32,032,000 B. t. u., the purchaser is required to pay for 56 pounds of water at the same rate per ton as for "dry coal," but as this percentage of moisture in average deliveries is inherently a constituent of the coal, it is considered as part and parcel of the coal by both the purchaser and the seller. If the coal delivered contains 3.5 per cent moisture, to procure 32,032,000 B. t. u., "dry coal," the purchaser has to pay 1 per cent more for coal because of the excess water above the normal amount, whereas if the coal contains 1.5 per cent moisture, the purchaser pays 1 per cent less for water. As the variations in moisture content, 1.5 or 3.5 per cent, are largely accidental, the season of the year being partly responsible for them, it is equitable that the purchaser and seller should share the uncertainty. The purchaser justly has a right, however, to demand that the seller shall guarantee a maximum moisture content, as a means of enabling the purchaser to compare one coal with another, as a guarantee that the seller will observe precautions against delivering coal that is unduly wet, and as a basis for adjusting the price of exceptionally wet coal.

The United States Weather Bureau in its annual reports gives the total precipitations per month and the maximum in 24 hours for different sections of the United States. At Washington, D. C., the greatest rainfall for any one month during the year ended December 31, 1910, was 5.89 inches, and the maximum for 24 consecutive hours was 3.67 inches.

As an example of the effect of a heavy rain on a car of coal in transit, a precipitation of 3 inches of water on a loaded 50-ton car, area of top about 360 square feet, would increase the weight of the coal 5.01 per cent, provided none of the water drained out or evaporated. It is obvious that if this coal is weighed and delivered immediately, special samples for moisture determinations should be collected and prepared at once and sent to the laboratory, as a basis for equitable adjustment of payment on account of the excessive amount of water in the coal. As the weight of the coal was increased by the excess water, there should be a corresponding decrease in the price to be paid.

If a railroad car or wagon so rained on should not be unloaded immediately after weighing and special moisture samples should not be properly collected, prepared, and sent hermetically sealed to the laboratory, it is obvious that the purchaser would pay a higher price than warranted, especially if the car or wagon stood for some time before sampling and some of the water drained out. Further, if the coal was not immediately unloaded and sampled or if the car continued in transit after weighing, then the coal at the top would soon dry; and in either case the effect of the 3-inch rainfall,

as indicated by the analysis, might be only a fractional percentage of the moisture contained in the coal at the time of weighing.

The determination of the moisture of coal delivered from stock piles is often of great importance, for the proportion of moisture contained in the small sizes, which are most abundant near the center of a stock pile and which absorb the rains, and melting snows in districts of heavy snows, may be from 10 to 15 per cent higher than when stocked. It is apparent, therefore, that special moisture sample determinations are necessary for equitable adjustment of payment on account of excessive moisture in coal which is stocked in piles exposed to the weather.

The specifications provide for the collection of "special moisture samples" if, in the opinion of the Government officials sampling it, the delivery contains moisture in excess of that guaranteed by the contractor. The "special moisture samples" are prepared in a manner to minimize moisture losses and may be taken and prepared independently of the gross samples collected for the determinations of heating value (B. t. u.), ash, and other specified data. If the analysis of the special sample shows a moisture content in excess of the contractor's guaranty, a proportionate deduction is made from the price to be paid for the coal.

"Special moisture samples" representing coal as delivered should be preserved in such a way as to minimize moisture loss, because if the moisture content of the sample, as shown by analysis, is less than the actual moisture content of the coal at the time it is weighed, either because of the sample being nonrepresentative or because of subsequent moisture loss during the storage or the preparation of the sample, the price to be paid for the coal delivered is correspondingly increased and an injustice is done the purchaser. It is equally important to protect the sample during collection or storage from being wet by rain, snow, or water from any other source in order to insure the sample representing the delivered coal. Evidently, therefore, particular attention should be given to procuring for the laboratory a sample containing the same percentage of moisture as the delivered coal.

To minimize moisture loss, the sample must be placed in a tight container and stored in a cool place before it is reduced. Moreover the sample should be reduced and transmitted to the laboratory as soon as practicable. It is well to remember that a sample taken from coal that has been exposed to a relatively high temperature, as coal from a boiler or furnace room, or from adjacent bins, has lost moisture and does not represent the coal as delivered. For the same reason samples should not be stored in the boiler room.

If the contractor does not guarantee a moisture content lower than can be actually maintained on an average, the collection of "special

moisture samples" and the making of price corrections on account of excessive moisture will seldom be necessary. Thus, the placing of heating value (B. t. u.) on the "dry coal" basis, with a maximum moisture content, eliminates frequent corrections of price on account of uncertain moisture variations.

ASH AND HEATING VALUE.

The heating value of coal from the same mine or from a group of mines operating the same bed of uniform character is directly influenced by the percentage of ash. By ash is meant earthy matter and impurities that do not burn. As the proportion of ash inherent (sometimes termed "intrinsic ash") in the coal substance as a rule does not vary widely in coal from the same bed, the heating value is largely determined by the amount and character of the free (sometimes termed "extraneous") impurities, and hence the manner in which coal is cleaned and prepared at the mine may greatly affect the heating value of commercial shipments. In collecting samples, therefore, it is of utmost importance that the samples should contain within reasonable limits the same proportion of impurities as the delivered coal from which they were taken.

Since the specifications provide for adjustment of price according to the ash content or the heating value, or both, it is evident that samples taken from delivered coal should show, by analysis, practically the same ash content and heating value as the delivered coal. In other words, the sample should contain the same proportion of slate, bony coal, pyrite and other impurities as the quantity delivered, in order that the price adjustment based on the analyses may be fair to both purchaser and seller. Obviously, it is important to guard against any accidental admixture of sand, cement, clinkers, or other foreign matter, while the sample is in storage or while it is being prepared for transmittal.

In this connection, attention is called to the fact that the gross sample should contain the same proportion of lump and fine coal as the delivery, for it has been determined that fine or slack coal may have an ash content, or a heating value, different from lump coal from the same shipment.

VOLATILE MATTER.

The volatile matter[a] of coal, as shown in analyses, generally contains some inert noncombustible matter, the proportion of which may range from 1 to 15 per cent. The character of the volatile matter in any given coal and the temperatures at which it is given off bear directly on the design and operation of furnaces for burning the coal efficiently and without smoke. The proportion of combus-

[a] Porter, H. C., and Ovitz, F. K., The volatile matter of coal. Bulletin 1, Bureau of Mines, 1910.

tible and noncombustible constituents in the volatile matter and the character of the combustible constituents, differ in different coals, and, therefore, the heating value of a coal can not be determined from a proximate analysis. Moreover, different coals with the same proportion of volatile matter may not behave alike in the furnace. In order to determine the comparative value of two coals for the same purpose, it is important to know both their chemical composition and their heating value. Hence the specifications recommended by the Bureau of Mines provide that before final awards of contracts, practical service tests may be made to determine the relative suitability of the coals offered.

In the specifications the contractor is required, in addition to giving the name and the location of the mine or mines producing the coal and the name of the coal bed worked, to specify the volatile matter content and other proximate analysis determinations of the coal he proposes to furnish. The content he specifies becomes the standard of his contract, and delivery of coal with a different percentage of volatile matter, indicating the substitution of a coal other than that offered, may result in the coal being rejected and the Government purchasing coal in the open market (the contractor being charged with the difference, if any, in cost) or in the contract being canceled.

The Government does not consider any scheme of applying penalties on account of variations in volatile matter equitable, because the character of the volatile matter and its heating value are not necessarily indicated by the volatile matter determination, and because this determination is made by an empyrical method whereby duplicate determinations on the same sample may differ as much as 2 per cent in different laboratories, or, in fact, in the same laboratory. Accordingly, no corrections in price are made for variations in volatile matter.

The bureau has received copies of commercial and municipal specifications drawn up apparently without regard to these facts. One specification, for instance, provides for a correction of 2 per cent in price for each 1 per cent of volatile matter in excess of the standard guaranteed. On a coal sold at $3 per ton, a deduction of 12 cents per ton for an apparent excess of 2 per cent in volatile matter might be made, when, in fact, the actual volatile matter of the coal was not above the standard. Specifications of this nature furnish legitimate grounds for opposition from coal companies and do much to discredit the specification method for the purchase of coal.

SULPHUR AND CLINKER.

Sulphur is commonly present in combination with iron or other elements. For a long time it was thought that the sulphur formed clinker,

but recent investigations point to the fact that sulphur is not the only cause of clinkering; in fact, there may be no difficulty from clinker in burning coal containing as much as 5 per cent or more sulphur. The relative proportions of iron, sulphur, lime, alumina, silica, etc., in the ash affect its fusibility, whereas the method of firing and the rate of combustion are important factors in the formation of clinkers. The exact relation of clinkering to the constituents of the ash is not known so well that one can definitely predict from an analysis of the ash whether a coal will or will not clinker. At many power plants the fireman slices the fire too often and works the ash up from the grates into the hot coal bed, where it melts and fuses into heavy dense masses of clinker. At high rates of combustion the ash in a given coal may clinker (though at lower rates it does not), because of the ash being raised to the fusing temperature. The fact that the percentage of sulphur does not necessarily indicate the behavior of the coal in the furnace is recognized in the specifications recommended by the Bureau of Mines. The contractors, however, are required to specify the sulphur content so that standards for the coals to be delivered may be established.

Many commercial and municipal specifications exact penalties for a slight increase in the sulphur content, although coal of high sulphur content may clinker less and its heating value may be higher. One specification received by the Bureau of Mines exacts a deduction of 5 per cent in price for each 1 per cent of sulphur in excess of the standard. In the case of a coal supplied at $3 a ton, the presence of 1 per cent of sulphur in excess of the standard, as indicated by analysis, would often result in an unwarranted deduction of 15 cents a ton. Under such a contract, the variations which it is recognized exist in sampling and analysis may cause a contractor unjustly to suffer a heavy deduction.

Both the sulphur and the volatile-matter content should be used to classify coals and to identify the coal guaranteed. Variations indicating the substitution of an unsatisfactory coal should be considered cause for rejection of the coal or for cancellation of the contract.

NEED OF EXPERIENCE AND CAUTION IN SAMPLING.

Persons without experience generally select a sample better than the average run of the coal delivered. Occasionally, a lump unusually free from layers of slate and impurities, and representing the best coal in the lot rather than the average, is selected. After being broken it is shipped to the laboratory in a cloth sack, so that it loses moisture during transit. The analysis of such a sample necessarily indicates a value higher than that of the coal delivered. As a quantity of coal

may vary greatly in composition, containing not only what may be termed coal proper with certain more or less constant impurities, but also slate, pyrite, and bony coal, a lump may be anything from almost pure coal to material without fuel value. Hence the analysis and test of a single lump may indicate a composition greatly different from that of the coal at hand. It is well to remember that, since the larger lumps of coal roll down and collect near the bottom of a pile or load, a sample taken entirely from near the floor does not always fairly represent the whole.

In spite of every precaution taken to prevent loss of moisture during the collection, preparation, and analysis of samples it is certain that loss of moisture may occur; also there may be too little or too much slate, bony coal, or other foreign matter collected in what is otherwise a truly representative gross sample, so that the determination of the heating value or ash content does not strictly agree with the actual value of these factors in the coal delivered. However, an experienced collector, by using good judgment and following the general directions given for collecting and preparing samples, can obtain samples so fairly representative that the results of the analyses are reasonably accurate. The suggestions that follow are presented for the guidance of those who wish to send representative samples to a laboratory for analysis and heating-value tests.

DIRECTIONS FOR SAMPLING.

SIZE OF THE GROSS SAMPLE.

The number of pounds to be taken as a gross sample to represent a given lot of coal varies with the character and condition of the coal, and not the amount of coal to be sampled. The character and proportion of the bony coal, slate, etc., and the size of the particles of both coal and impurities are the governing factors. It is therefore evident that sampling should not be left to an inexperienced person, but should be done by one who is thoroughly familiar with the significance of these factors and has some knowledge of the coal to be sampled.

There is greater probability of taking too little than of taking too much coal for a gross sample. It is generally true that the larger the gross sample the less is the chance of its being nonrepresentative. Large samples must be taken in sampling coals that carry a varying proportion of large pieces of slate, bony coal, or pyrite, for it is evident that including or excluding large pieces of the impurities must affect the analysis of a small sample considerably; hence, the analysis may not indicate the quality of the coal sampled and, consequently, may be worthless for determining the price to be paid for

the coal. If the free impurities are small particles and are well distributed throughout the coal, representative samples may be obtained easily. As stated by Bailey: [a]

> The larger the size of the pieces of impurities the larger should be the original sample to prevent excessive error, and the same is true with respect to the quantity of impurities.

The Bureau of Mines has carried on experiments and tests to determine the quantity of coal necessary in gross samples to make them fairly representative of the coal sampled. The table following shows the results of one series of tests and brings out very clearly the necessity of gross samples larger than those generally collected.

Increase in accuracy of B. t. u. determination with increase of gross sample.

Weight of sample (pounds).	B. t. u. "dry coal" differences.												Number of B. t. u. determinations involved in arriving at the average difference.
	Test.										Average.	Maximum.	
	a	b	c	d	e	f	g	h	i	j			
100	158	301	38	190	620	10	325	40	365	459	251	620	20
200	132	173	40	125	525	130	220	65	285	305	200	525	40
300	120	59	18	177	410	17	92	43	135	179	125	410	60
400	86	65	55	140	283	62	89	45	134	151	111	283	80
500	0	67	81	156	244	16	41	20	61	67	75	244	100
600	61	97	86	208	159	62	12	22	34	36	78	208	120
700	54	67	55	153	88	56	2	12	10	13	51	153	140
800	35	62	76	102	107	49	27	9	36	27	53	107	160
900	17	28	59	85	124	69	2	23	25	11	44	124	180
1,000	30	19	34	49	94	59	8	35	27	11	37	94	200
1,100	42	26	26	34	51	64	1	29	28	16	32	64	220
1,200	18	8	15	.....	70	53	21	30	51	26	32	70	216
1,300	7	5	22	.....	55	45	41	15	56	2	25	56	234
1,400	.....	.....	.....	.....	.....	51	.....	.....	.....	.....	.....	51	.....
1,500	.....	.....	.....	.....	.....	38	.....	.....	.....	.....	.....	38	.....

The table shows the differences obtained by two different collectors on samples weighing 100 pounds, 200 pounds, 300 pounds, etc. On test (a) the collectors first took 100-pound samples which showed a difference of 158 B. t. u. in heating value. Each man then collected 100 pounds more. The heating values of the first and second 100-pound samples were averaged for each collector and the difference between the average results of the two collectors for two 100-pound samples was 132 B. t. u. Each man then collected 100 pounds more of coal. The averages of the two collectors for the three 100-pound samples differed by 120 B. t. u. In like manner the difference in heating value for each successive 100 pounds of coal was determined. Ten (a to j) similar tests were made. For the cargo, test (a), each collector took thirteen 100-pound samples of coal, or 1,300 pounds, to represent the cargo, and the average of the 13 B. t. u. determina-

[a] Bailey, E. G., How to sample coal and coke, Fuel Testing Co., Bull. 4, 1910. 32 pp.

tions for the 13 samples of one collector differed by only 7 B. t. u. from the average of the 13 B. t. u. determinations of the other collector.

The average differences are of especial significance as showing the effect of the size of the sample as a factor in obtaining a heating value which truly represents the coal sampled, because the variations that are recognized to exist in the reduction of gross samples and in the analytical work were minimized by averaging; for example, in determining the average difference of 37 B. t. u. for the 1,000-pound sample, 100 heating-value determinations were averaged for each collector, 200 such determinations being involved in obtaining the

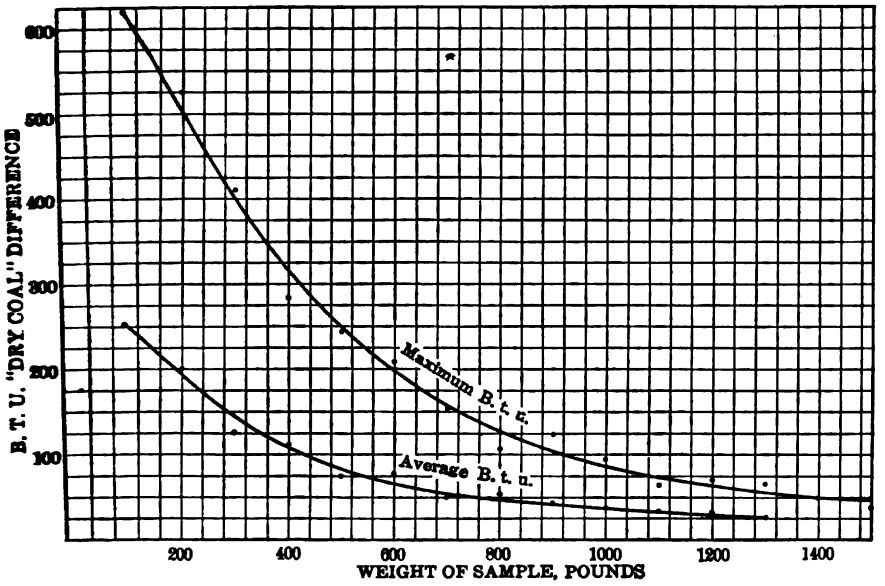


FIGURE 1.—Curves showing how the heating-value determination becomes more representative with increase in size of gross sample.

difference of 37, and this difference of 37 B. t. u. may be considered as representing the actual difference in the heating values per pound of the two 1,000-pound gross samples.

The results given in the table have been plotted in figure 1, the heating-value differences being used as ordinates and the weights of the samples as abscissas. The differences in ash, "dry coal," of the samples are also plotted (fig. 2), the average differences in the percentages being used as ordinates and the weights of the samples in pounds as abscissas.

The curves show very strikingly the effect that the size of the gross sample has in insuring representative results. As the size of the sample is decreased the differences between separate samplings become

greater and greater—extending the curves toward the left will show that with very small samples the difference may become equivalent to the difference between pure coal and "slate." On the other hand, the curves show that as the weight of the sample is increased the differences of ash and heating value rapidly decrease until a point is reached where the accuracy obtained by taking larger samples would not ordinarily be warranted, because of the expense which would be involved in making special provisions for handling the very large samples.

The large differences with the small samples show very forcibly the necessity for collecting large gross samples to insure the results of analyses and heating value tests being reliable for determining the price to be paid for delivered coal and for preventing injustice

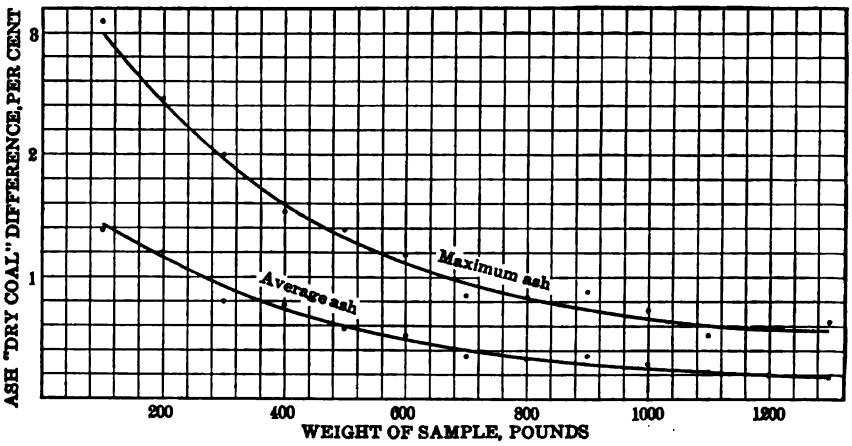


FIGURE 2.—Curves showing how the ash determination becomes more representative with increase in size of gross sample.

to either seller or buyer. The collectors each endeavored to collect 100 pounds of coal representative of the quantity sampled, but with samples weighing only 100 pounds a maximum difference of 620 B. t. u., "dry coal," resulted. This difference was not due so much to any difference in the actual heating value of the coal substance as to a collector unintentionally and by chance including or excluding pieces of slate, bony coal, pyrite, and other impurities in taking his sample. Obviously payment based on either one of two samples differing 620 B. t. u. in heating value would probably have worked to the disadvantage of either the seller or the buyer.

One should remember that the differences given in the table for the samples as they grow larger decrease, not only because of the fact that the samples gradually approach each other in homogeneity, but because the variations in analytical work and in the reduction of the samples are diminished as the number of determinations used in

obtaining the averages increases. The maximum difference of 620 B. t. u. for the 100-pound samples is the difference between two single determinations, each of which includes the variations due to laboratory work and the reduction of the samples. In this case, if the samples had again been reduced and analyzed, variations due to laboratory work and sampling might have given a difference greater or less than 620 B. t. u. The maximum difference of 94 B. t. u. for the 1,000-pound sample is the difference between the average of 10 heating-value determinations for one collector and the average of 10 determinations for the other collector. The laboratory and sampling variations in this case are, it is seen, averaged or divided, and it is obvious that the effect of these variations is greatly less in this case than in that of the two single determinations which differed by 620 B. t. u.

As has been stated and as is indicated by the curves, the differences in ash and heating value rapidly decrease as the size of gross sample is increased until a point is reached where the accuracy obtainable by taking larger gross samples is affected but slightly. The table below shows the results of a series of four tests in which each collector took samples of 2,500 to 4,450 pounds to represent cargoes of approximately 5,500 tons each:

The effect of size of gross sample on heating-value determination.

Test.	Weight of gross sample.	Number of samples.	B. t. u., "dry coal."						Difference between averages.	Number of B. t. u. determinations involved.
			Regular.			Experimental.				
			Originals.	Duplicates.	Average.	Originals.	Duplicates.	Average.		
	Pounds									
A	2,500	5	14,656	14,652	14,654	14,642	14,666	14,654	0	20
B	4,450	8	14,722	14,743	14,733	14,771	14,770	14,771	38	32
C	3,920	14	14,802	14,809	14,806	14,848	14,856	14,852	46	56
D	3,640	13	14,876	14,887	14,882	14,805	14,840	14,823	59	52

One object of these tests was to establish the fact that beyond certain limits accuracy is little affected by larger gross samples, and on comparing the differences in this table with the differences in the table on page 16 it is seen that this fact is borne out.

In this table the weight of the gross sample, in test A, for instance, is the quantity of coal which each collector took to represent the cargo. Each collector's gross sample was reduced to five successive quantities—that is, in test A, after the collector had accumulated 500 pounds, he prepared it for transmittal to the laboratory. After each of his 500-pound samples had been reduced to approximately 6 pounds, it was halved by the use of a riffle and the halves placed

in separate mailing cans. One of these halves, termed the "duplicate," is ordinarily retained and held as a reserve, while the "original" is mailed. In this investigation, however, the "duplicates" were mailed as well, and the "originals" and "duplicates" were both analyzed. Then in test A the average of the analyses of the five "original" samples was 14,656 and of the five "duplicates" 14,652, the average of the "originals" and "duplicates" being 14,654. The samples collected by one collector are termed the "regular" samples, while the samples collected by the other collector are termed the "experimental."

In tests A, B, and C the collectors sampled the same cars, portions of coal being taken from every second car of coal discharged into the ship. In test D one collector took portions from the first, third, fifth, etc., cars, while the other collected portions from the second, fourth, sixth, etc., cars, so that their combined gross samples of 7,280 pounds was made up of coal from every car loaded. As the collectors sampled entirely different cars in test D, the fact that the difference between the heating values of their samples is slightly greater than on the other tests is not surprising, and shows by comparison with tests A, B, and C that the method of sampling every second car gives results which can be taken as representing an entire cargo.

The differences obtained between the "originals" and "duplicates" vary from 1 B. t. u., for the "experimental" samples of test B, to 35 for the "experimental" samples of test D, which shows that there is an element of chance which sample is forwarded and termed the "original." In the routine reporting on cargoes only one set of samples is analyzed to represent a cargo; thus in test D it may be considered that the value which might have been reported on the cargo would have been either 14,876, 14,887, 14,805, or 14,840, depending on which set of samples had by chance been forwarded for routine analysis and report—that is, a difference of 82 B. t. u. in the reports may have resulted. Differences due to such causes, as well as to others, must be recognized in interpreting a coal analysis and in drawing up specifications that provide for adjustment in price according to the determinations reported by the laboratory. These determinations reported are given a liberal construction in the specifications prepared by the Government (see pp. 52–66).

The tests were made with Pocahontas and New River coals. These coals are relatively free from the "extraneous" impurities, and are comparatively easy to sample. Greater differences are to be expected with coal of firmer physical structure containing a considerable proportion of impurities in large pieces or large lumps of coal with strata of bone, slate, pyrite, or sulphur balls or lenses.

Hence larger gross samples should be collected from such a coal to insure the results of analyses and tests being fairly representative of the coal delivered.

Because small samples may, and quite generally do, indicate a value for the coal sampled that differs from the actual value, the specifications that are recommended by the Bureau of Mines provide for the collection of a gross sample weighing not less than approximately 1,000 pounds. If the coal to be sampled contains an unusual amount of impurities, and if the pieces of such impurities are very large, it may be necessary to collect larger gross samples, of even 1,500 pounds or more.

Whether a delivery consists of 1 ton or 100 tons, the need of the gross sample weighing 1,000 pounds or more is the same. One such sample carefully taken and reduced to a quantity convenient for transmittal to the laboratory can well represent a delivery of several hundred or a thousand tons of coal.

For convenience, however, and to avoid long storage of the samples, the Bureau of Mines considers it advisable that one sample of 1,000 pounds or more should be collected each week in the case of Government contracts calling for more or less regular deliveries each week. If, however, the quantity delivered during a week is relatively small, then the sample may represent the coal delivered during a longer period. To facilitate accounting, many branches of the Government service order contractors to deliver certain quantities, usually 100 to 500 tons. In such cases the samples are collected to represent the order without special regard to the period covered, and one or more gross samples of 1,000 pounds or more each is collected, as may be most practical and expedient in view of the facilities for sampling and the other considerations involved.

In sampling cargo deliveries of 5,000 and more tons, the Bureau of Mines collects from 4,000 to 5,000 pounds of coal as a gross sample. In order that the preparation of the samples may proceed while the cargo is being loaded, after approximately 500 pounds has been collected it is reduced down to a quantity convenient for mailing to the laboratory, and each succeeding 500 pounds is likewise reduced. This procedure makes unnecessary the accumulation of a quantity of coal that can not be systematically and conveniently handled in the short time and the small space usually available. Two or more of the samples are combined and reduced to one in the laboratory, and four or five analyses are usually made for a cargo, and a report on the cargo is obtained by averaging the analyses. The samples may, however, be mixed in the laboratory and only one analysis made to represent the cargo. Though the experiments which have been made indicate that a sample of approximately 1,000 pounds will give results fairly representative of the cargo, the objection to

the 1,000-pound sample is that it is too small to allow of the frequent collection of shovelfuls or portions of any quantity throughout the loading of the cargo. As it generally happens that coal from a number of mines is loaded into the same cargo, it is desirable to collect a considerable quantity of coal, so that each mine may be well represented in the gross sample. It is obvious that the more frequently the portions are collected and the greater the quantity sampled the less the probability that the sample will be nonrepresentative; accordingly, the bureau considers that in sampling 5,000-ton cargoes safety lies rather in the larger gross sample.

WHEN TO COLLECT SAMPLES.

The best opportunities for procuring representative samples are afforded while the coal is being unloaded from railroad cars, ships, and barges, or while it is being dumped from wagons. Once the coal is stored in piles or bins, or loaded on vessels, the procuring of representative samples is practically impossible unless the whole quantity of coal is immediately handled again and the conditions for sampling become favorable. Samples collected from the coal exposed in piles, bins, barges, or ships can be considered representative only under the condition that the mass of coal is homogeneous throughout. Such a condition is highly improbable and uncertain, and the analysis of samples collected from the surface may give results that are very unreliable as indicating the nature of the entire quantity, and that may be worthless as a basis for determining an equitable price to be paid for the coal.

COLLECTION OF GROSS SAMPLES.

When coal is being unloaded from wagons, railroad cars, ships, or barges, a shovel or a specially designed tool may be used for taking portions or increments of 10 to 30 pounds to make up the gross sample of coal. As the size of the increments should be governed by the size and weight of the largest pieces of coal and impurities, increments of more than 30 pounds may be required for coals containing large pieces of coal and impurities.

If one chute or conveyer is used for delivering a considerable quantity of coal to or from wagons, cars, or ships, it may prove expeditious and economical to devise a mechanical means for collecting portions from fractional parts of the discharged coal, or continuously deflecting a portion of the coal as it falls down the chute, or diverting from the conveyer definite portions of coal, and thus mechanically and automatically collecting the gross sample.

The mechanical collection of samples is preferred to shovel sampling, as it eliminates the personal equation. As the mechanical

sampler does not discriminate for or against taking more or less "slate" or other impurities, so a person should collect samples with a shovel in the main without regard to impurities, because the amount of the impurities included in a sample must largely be left to chance, since it is impossible to rate correctly the proportion of the impurities concealed in the coal, however competent the sampler may be. A mechanical sampler should preferably take the whole of the stream of coal flowing down the chute a part of the time rather than a part of the stream all the time, because the sizes and character of the pieces of coal and impurities are not apt to be evenly distributed across the stream. Excellent opportunity is afforded for procuring representative samples if the entire consignment of coal is crushed immediately after it is weighed and delivered, for then the samples can be collected from the crushed coal. If the coal is conveyed from the crusher by a conveyer, means can be devised for mechanically and automatically diverting from the conveyer definite portions of coal to make up the gross sample.

The portions should be regularly and systematically collected, so that the entire quantity sampled will be represented proportionately in the gross sample. The interval at which the portions are collected should be regulated so that the gross sample collected will weigh not less than approximately 1,000 pounds. If the coal contains an unusual proportion of impurities, such as slate, bony coal, and pyrite, and if the pieces of such impurities are very large, it will be necessary to collect gross samples of even 1,500 pounds, or more, but for slack coal and for small sizes of anthracite, if the impurities are not in abnormal proportion or in pieces larger than about three-quarters of an inch, and if the impurities are evenly distributed throughout the coal, a gross sample of approximately 600 pounds may prove sufficient. The gross sample should contain the same proportion of lump coal, fine coal, and impurities as the coal delivered. As the portions are collected they should be deposited in a receptacle having a tight-fitting lid provided with a lock.

WAGONLOAD DELIVERIES.

A gross sample taken by hand from coal delivered by wagon at a Government building should consist of shovelfuls of coal taken from every first, second, or third wagonload as it is being discharged, the number of shovelfuls taken and the loads sampled being dependent on the number of loads which the gross sample is to represent. If the coal is discharged immediately into a crusher, it is preferable to collect shovelfuls of the crushed coal.

If it is desired to sample coal delivered in small lots by wagons to a large number of buildings from one source of supply where the coal

is all weighed; and if the collection of samples at each building will involve considerable expense, samples may be conveniently collected at minimum expense at the source of supply when the coal is loaded into the wagons. In such case the samples are collected without regard to delivery points, the object being to sample the coal loaded into the wagons. At the end of the month or upon the completion of an order the corrected price per ton, if there be any corrections on account of coal varying from the standards guaranteed by the contractor, applies to each individual delivery point. Shovelfuls of coal should be taken from each wagonload, or every second or third, etc., wagonload, the number of shovelfuls taken and the number of wagonloads sampled being dependent upon the number of loads which the gross sample is to represent.

It is important to obtain representative portions of coal from every part of the delivery, so that the sample will show the quality of the delivery or order as a whole. The gross sample should contain about the same proportion of lump coal, fine coal, and foreign matter as the coal delivered.

CARLOAD DELIVERIES.

Samples taken from railroad cars should not be limited to a few shovelfuls of coal procured from the top of a car, for the size of the coal and the proportion of foreign matter may vary from the top to the bottom of the car. The only way to obtain a representative sample is to take a number of shovelfuls or portions of coal from different points in a car, from top to bottom and from end to end, while the coal is being unloaded.

When sampling coal delivered from open cars and unloaded by hand shovelfuls should be taken at regular intervals as the coal is unloaded into wagons or bins.

When coal is being sampled from dump cars, ladlefuls (see p. 25) or shovelfuls should be taken from the stream of coal being discharged to the bins or ship. If the discharged coal is immediately crushed the gross sample should preferably be collected after the coal leaves the crusher.

If a number of carloads are delivered on an order or during a short period the preparation of a gross sample of 1,000 pounds or more for each car would involve considerable time, labor, and expense at the delivery point, as well as in the laboratory; in fact, a gross sample for each car would not be required, for if a gross sample is carefully taken and prepared it can well represent a number of cars. If a gross sample is collected to represent two or more cars representative portions of the coal should be taken in equal quantities from each car.

The method followed by the Bureau of Mines in sampling coal discharged from railroad cars directly into ships is given under "Cargo shipments."

CARGO SHIPMENTS.

In sampling cargoes, as in sampling carloads, portions of coal should be taken in equal quantities and at frequent and regular intervals so as to represent proportionate parts of the consignment as a whole, either while the coal is being loaded or unloaded. There is no assurance that a sample or a series of samples taken from the top of the cargo represents the cargo as a whole; in fact, it is very doubtful if such samples are ever representative. If the vessel is being loaded directly from the railroad car portions of coal should be taken from as many cars as practicable; for cargoes of 4,000 tons and more loading in 12 to 36 hours it may be impossible to collect portions from every railroad car dumped, unless a number of sample collectors are employed or the sample is collected automatically by a mechanical device, because of the rapidity with which the cars may pass over the pier. In this event portions should be taken from every second car if practicable and these portions mixed to form gross samples.

The Bureau of Mines has charge of the sampling of cargo shipments of coal to the Isthmus of Panama, account of the Panama Railroad Co. The coal is loaded at present from piers on Hampton Roads, Norfolk, and Newport News, Va., directly from railroad cars into ships carrying from 4,000 to 6,500 tons of coal. The method followed is to collect portions of coal from every second railroad car dumped, from 60 to 70 pounds being taken from each car sampled. From 4,000 to 5,000 pounds are usually collected as gross samples to represent a cargo (see p. 21 under "Size of the gross sample"). The gross samples are reduced in successive stages, rather than accumulated and later reduced to quantities convenient for transmittal to the laboratory.

The portions making up the gross samples are taken from the coal as it is discharged from the bottom-dump cars by the use of a shovel or a specially constructed ladle. The ladle has a handle about 5 feet long and a bowl 1 foot in diameter at the top, 9 inches at the bottom, with depth of 9½ inches, and holds from 25 to 30 pounds of coal. To collect samples with the ladle, it is rested on the rails of the track or on the chains that support the gates of the car. Two ladlefuls are usually taken from a car, one from one side and the other from the other side at the opposite end, thereby obtaining portions from different parts of the car. Care is observed to not collect portions of the first or last coal spilling from the car, for the moisture content or proportions of foreign matter in such coal may render the sample unrepresentative. If circumstances permit, the ladle is shifted and filled from different sections of the stream of falling coal. It sometimes happens that the coal is discharged very rapidly, in which case the collector must be alert to collect portions at the most desirable

periods and must take care that the coal does not wrench the ladle out of his hands or jerk him down and do him bodily injury.

The method of sampling cargo shipments has been established as the result of numerous experiments, and it is considered that the analytical reports obtained are representative within limits reasonably to be expected for work of this nature. In the table below are given the results obtained by two men (in two cases, three) separately and independently collecting samples to represent the same cargo. Under test A, three men collected separately and independently 13 100-pound samples each, and the average of the analyses of the 13 samples of the man regularly delegated on the work is indicated as the "Regular" result, while the results of the other two are designated "Experimental." The first differences under column headed "Difference" are between the "Regular" and first "Experimental," and the second differences are between the "Regular" and second "Experimental," while the third differences are between the two "Experimentals" (for all practical purposes of comparison, one of the "Experimentals" being considered as "Regular" in obtaining the third differences).

The samples were collected with shovel or ladle, as indicated by S or L, one object of the tests being to determine relative values of the two tools for use in collecting samples—a study of the table shows that as far as results are concerned, it is immaterial which is used.

All the samples of certain sets were analyzed in the Bureau of Mines laboratory at Pittsburgh, Pa., the others being analyzed in its Washington laboratory. The samples analyzed in Pittsburgh are indicated by P, those in Washington by W. A study of the results show that for practical purposes, it is immaterial in which laboratory the determinations were made, especially with respect to the ash content and the heating value. The laboratories differ somewhat, however, in the moisture determinations, this being due to the fact that when the experiments were conducted, the Washington laboratory did not follow the "air-drying" method used at Pittsburgh for determining moisture. The methods are diagrammatically shown on pages 40 and 41. The results show, as expected, that the "air-drying" method gives a slightly higher moisture content.

In tests A to F, inclusive, each man sampled every third car, collecting about 35 pounds to the car, three such portions, or 100 pounds, representing every 9 cars loaded into the ship. The men sampled the same cars, except those indicated by asterisks; in which cases the "Experimental" samples were collected from cars not sampled by the other man or men.

In tests G, H, I, and J, the men collected approximately 70 pounds from every second car, 8 such portions being mixed, giving from 500

to 550 pounds to represent 16 cars loaded into the ship for tests G and H, whereas 4 such portions were mixed for tests I and J.

For practical purposes, the table shows that it is immaterial whether the shovel or ladle is used; whether samples are analyzed in Washington or Pittsburgh laboratories; that the larger gross samples do not give closer agreement than the smaller ones; and that portions of coal collected from every second or third car give results which fairly represent a cargo.

Check cargo sampling.

Test.	Number of samples.	Weight of gross samples.	Regular.			Experimental.			Difference.		
			Moisture "as received."	Ash, "dry coal."	B.t.u. "dry coal."	Moisture "as received."	Ash, "dry coal."	B.t.u. "dry coal."	Moisture "as received."	Ash, "dry coal."	B.t.u. "dry coal."
		Pounds.	*Per cent.*	*Per cent.*		*Per cent.*	*Per cent.*		*Per cent.*	*Per cent.*	
A	13	1,300	(W) (S) 2.52	4.61	15,017	(P) (S) 3.27	4.53	14,976	0.75	0.08	41
						(W) (S)* 2.45	4.58	15,032	.07	.03	15
									.82	.05	56
B	13	1,300	(W) (L) 1.85	6.92	14,710	(P) (S) 2.14	6.76	14,715	.29	.16	5
						(P) (L)* 2.18	6.54	14,717	.33	.38	7
									.04	.22	2
C	13	1,300	(W) (L) 1.71	7.06	14,664	(P) (S) 2.13	6.64	14,686	.42	.42	22
D	11	1,100	(W) (L) 3.09	4.85	14,992	(W) (S) 3.07	4.59	15,026	.02	.26	34
E	13	1,300	(W) (L) 3.09	6.43	14,742	(W) (S) 3.28	6.98	14,687	.19	.55	55
F	15	1,500	(W) (L) 2.08	6.63	14,760	(P) (S) 2.47	6.57	14,722	.39	.06	38
G	5	2,500	(W) (L) 1.72	6.70	14,654	(W) (S) 1.67	6.96	14,654	.05	.26	0
H	8	4,450	(W) (L) 2.23	6.57	14,733	(W) (S) 2.28	6.34	14,771	.05	.23	38
I	14	3,920	(W) (S) 2.64	5.70	14,806	(W) (S) 2.45	5.42	14,852	.19	.28	46
J	13	3,640	(W) (S) 3.49	5.15	14,882	(W) (S)* 3.65	5.53	14,823	.16	.38	59
Average difference									.27	.24	29

The bureau is making a study of the conditions on the piers, with a view of devising means whereby the samples can be automatically and mechanically collected, prepared, and reduced to quantities convenient for transmittal to the laboratory. At one of the three railroad terminals loading at Hampton Roads, Va., the bureau has installed a crusher for crushing the gross samples to ¼-inch mesh and finer, and parting devices for reducing the samples in quantity. It is proposed to install crushers and equipment at the other two points.

Because of the suddenness with which the coal may break through and drop out of the railroad car after the gates are open and because of the momentum of the rapidly falling coal, it may be impossible to collect a satisfactory sample by attempting to catch coal in a shovel or ladle. In such event, it may be necessary to collect shovelfuls of

the coal that has overflowed on the pier. If beams 10 to 12 inches wide span the pockets immediately underneath the car, a fairly satisfactory sample can often be collected in shovelfuls from the coal lodging on the beams.

Though the collector may use the shovel or ladle to the best of his ability, these tools may collect portions of coal that do not contain lump coal in the same proportion that exists in the car. Especially may this occur if the coal contains a large proportion of lumps of considerable size. In that event, the collector should collect portions of lumps from time to time and add them to the sample. Necessarily the collector must be relied upon to collect a sample under these conditions that will fairly represent the proportions of lump and slack coal contained in the coal sampled and it follows that the collector must have experience in sampling and be able to judge the coal and the sample.

In sampling large quantities of coal in a short period the collector frequently works under disadvantages and has not the opportunity to select more leisurely and methodically portions of coal to make up a gross sample, as he can in sampling wagonload deliveries or in single carloads or barges unloaded by shovel, clam shells, or grab buckets, and this fact should be considered in comparing the results of sampling cargo with wagon or railroad-car deliveries.

In collecting samples from coal as it is unloaded from a ship, the same general instructions apply; the cargo being systematically sampled during the entire period of unloading, so that samples will be collected which represent the quantity as a whole. If unloaded by grab buckets or into barrows, shovelfuls should be collected at regular intervals from the buckets or barrows. If the coal is crushed immediately after it is unloaded, it is preferable to collect samples from the crushed coal.

SAMPLE RECEPTACLE.

As the wagons or railroad cars may arrive irregularly and the coal be intermittently unloaded, a metal receptacle or wooden box of a size to hold a gross sample of at least 1,000 pounds, with a tight-fitting lid, which can be locked, is required for holding the portions of coal taken from each wagon or car until the gross sample is completed. In sampling cargo deliveries, buckets holding 60 to 70 pounds may prove more satisfactory to use for receiving the portions making up a gross sample, as the samples are usually worked down as the loading progresses and the buckets are convenient for carrying the coal to the space available for preparing the gross samples. If the gross samples are stored, a box should be provided or else a sufficient number of buckets with tight-fitting lids and locks should be available.

The buckets, boxes, or receptacles should be inspected each time before using and thoroughly cleaned to remove the coal dust remaining from previous samples and any foreign matter that may by chance be in them.

COLLECTION OF SPECIAL MOISTURE SAMPLES.

During the collection of gross samples and their reduction to quantities that are convenient for transmittal to the laboratory and that correctly represent the ash, sulphur, and heating value of the coal sampled, it is ordinarily impossible to retain in the sample all the moisture that was in the delivered coal. Obviously, if it is desired to determine the amount of moisture in delivered coal, special moisture samples are usually necessary. Owing to the fact that a sample unavoidedly loses moisture during every stage of handling and preparation, the special moisture samples must be, in a sense, grab samples and must be collected, prepared, and placed in a sealed container with as little delay as possible.

If a gross sample is collected during a period of a few hours or a very few days and the sample is collected and stored under conditions so as to preserve its moisture content, a special moisture sample may be collected from the gross sample after it has been rapidly crushed, so that it will all pass through a 1-inch screen (the fineness to which the gross sample is reduced by the first crushing, see table on p. 33). It should be collected in a place comparatively cool and protected from rain, snow, wind, and the sun's rays. A small scoop may be used for collecting the sample. The scoop should have a capacity to hold about one pound of coal—a scoop with bottom about 2½ inches wide and 8½ inches long and vertical sides about 2 inches high is about the right size. As the crushed coal is shoveled into a conical pile (p. 34), scoopfuls should be regularly and systematically collected so that approximately one scoopful will be collected to every two shovelfuls (about 30 pounds) deposited on the cone, thereby collecting a special moisture sample weighing from 30 to 50 pounds. As the scoopfuls are collected they should be placed in a receptacle which can be tightly closed. After the gross sample has been formed into a conical pile (p. 34) and the special moisture sample weighing 30 to 50 pounds has been accumulated, the special moisture sample should be immediately and rapidly crushed so that no pieces of coal or impurities are larger than one-half inch; and it should be rapidly coned, flattened, and quartered, and a mailing can (p. 43) filled by taking portions from each quarter by use of the scoop (each portion only partly filling the scoop). The mailing can should be properly sealed at once (p. 43) and forwarded to the laboratory.

If the gross sample in the above case is crushed mechanically and reduced to the quantity desired for transmittal to the laboratory by

use of sampling machines in such a manner and in machines so designed that moisture losses are minimized, then the collection of a special moisture sample may prove unnecesssry, as the moisture determination made on the regular sample will represent, within reasonable limits, the moisture content in the coal delivered.

If a gross sample is collected during a period of several days, and if it is manifest that the sample will lose moisture during its storage and preparation, the special moisture sample should be accumulated by placing in a receptacle small portions of the freshly taken increments making up the gross sample. Under these conditions it is desirable that a receptacle which can be hermetically sealed and has a capacity of about 70 pounds be provided in which to preserve the portions making up the special moisture sample as they are taken each day. After the gross sample is accumulated a quantity convenient for transmittal to the laboratory should be selected from the special moisture sample (in accordance with the method given above) and forwarded in the mailing can, properly sealed for special moisture determination.

If two or more railroad cars are to be represented by one gross sample, and if the cars contain different amounts of moisture seemingly in excess of the maximum moisture content guaranteed, moisture samples should be taken separately from each car. If a single gross sample is to represent several days' delivery, and if because of heavy intermittent rains there is a considerable difference in moisture content between each day's delivery, and each contains moisture in excess of the maximum content guaranteed, then a special moisture sample should be taken representing each day's delivery. Payment for the entire quantity on account of ash and B. t. u., "dry coal," is determined from the analysis of the gross sample, but corrections on account of excessive moisture should apply to the particular car or cars, or day's delivery. The purchaser would be at a disadvantage on account of heavy rain if corrections for moisture did not apply to each day's delivery or fraction of the delivery, because if the special moisture samples were mixed, and one special moisture determination only made, the effect of the heavy rains would largely be averaged out in the mixture, and, in addition, in storing, mixing, and reducing the samples more or less moisture would be lost.

PREPARATION OF THE GROSS SAMPLE.

Though a gross sample may be collected ever so carefully and may represent the coal sampled, unless it is prepared in accordance with well recognized principles, the results of analysis and test may be worthless for determining equitable settlement for the coal.

The portions taken in making up the gross sample should be immediately placed in a box or a receptacle having a tight-fitting cover

and lock for storage until it is reduced to a quantity convenient for transmitting to the laboratory. Each receptacle should be securely locked and the key held by a responsible employee.

If it is desired to determine the moisture content of a gross sample that has been collected in a comparatively short time under precautions to minimize moisture loss, the sample should be placed in a comparatively cool place and the crushing, mixing, and reduction for shipment to the laboratory should be done as rapidly as possible; this is imperative if the analysis of the sample is to represent the approximate condition of the coal with respect to its moisture content at the time it was weighed. Obviously, a sample on which a moisture determination is desired should be protected from rain and snow and strong air currents or winds and the sun's rays during its storage and preparation.

The proper preparation of a gross sample for shipment to the laboratory involves three operations: (1) Crushing, (2) mixing, and (3) reduction in quantity. The operations proceed in stages until the final sample is obtained.

The crushing may be done by a mechanical crusher or by hand with an iron tamping bar or sledge on a smooth, clean, sheet-iron plate, of suitable dimensions, or on a solid floor—in the absence of a sheet-iron plate or smooth tight floor, the crushing may be done on a heavy canvas—to prevent the accidental admixture of any foreign matter. The mixing and reduction may be done by hand with a shovel, or, mechanically, by means of riffles or sampling machines.

It is obvious that if the gross sample is reduced in quantity without crushing, as the sample becomes smaller, the effect of the selection or rejection of one or more of the large pieces of slate, or other impurities, in the portion of the sample retained multiplies rapidly. To illustrate: A sample of 1,000 pounds contains a piece of slate weighing 1 pound, which is one-tenth per cent of the weight of the gross sample. If the sample is halved in quantity without crushing, the half of the sample retained for further reduction will contain two-tenths per cent ash more or less than the rejected half, according to whether the piece of slate went into the retained half or into the rejected half. In halving the 500-pound sample, the 1-pound piece of slate would have an effect of four-tenths per cent on the ash content of either the retained or rejected half. If, in continuing the reduction of the sample by halving each time, the 1-pound piece of slate by chance should fall into the retained half on each successive halving, it would have an effect of 12.8 per cent on a $7\frac{13}{16}$-pound sample—that is, in halving the $15\frac{5}{8}$-pound sample, the 1-pound piece of slate would cause the ash content of one-half of the sample to be 12.8 per cent higher than the other; in other words, if the average ash of the coal is 10 per cent, one of the $7\frac{13}{16}$-pound samples would show an ash content of 22.8 per cent.

This fundamental principle that the weight of the largest piece of impurities should be relatively very small in ratio to the weight of the sample at each halving, is recognized in the instructions for sampling issued by the Bureau of Mines which specify that the sample should be successively crushed, mixed, and reduced.

In the sampling of ores, it is recognized that the particles of ore must be crushed to varying degrees of fineness in order to obtain results within an allowable limit of error, and to accomplish this, elaborate and costly plants are constructed and maintained. Brunton [a] states that experiments and calculations and a general consideration of the subject indicate that the size to which ore must be crushed for sampling, in order to come within an allowable limit of error, will depend upon:

1. The weight or bulk which the sample is to have. Evidently the smaller the sample the finer the material must be crushed.

2. The relative proportion between the value of the richest mineral and the average value of the ore. If the average grade of the ore is high, in comparison with the grade of the richest mineral, a particle of richest mineral of a given size and value will have less percentage effect on the sample than the same particle would have on the same amount of lower-grade ore; therefore, other conditions being the same, with high-grade ores we may crush more coarsely than with low-grade ones, and still keep within the same percentage of error; while if the richest mineral is of comparatively high grade, a particle of it, of given size, will have a greater effect on the sample than if it is of low grade, and this will necessitate finer crushing.

3. The specific gravity of the richest mineral. The higher the specific gravity of the richest mineral the greater the value contained in a particle of given size and grade, and hence the greater the influence of such particle on the sample; from which follows the necessity of keeping down the size of the largest particles by finer crushing than is required when the richest mineral is of lower specific gravity.

4. The number of particles of richest mineral which are likely to be in excess or deficit in the sample is evidently an important factor; a liability to a large number necessitating especially fine crushing. But such liability can result only from imperfect mixing, and for material mixed with average thoroughness this number must be small.

And in conclusion, this paper further states that:

The results of the investigations recorded in this paper show how absolutely necessary it is that ore samples should be recrushed after each successive "cutting-down," so that as the sample diminishes in weight, there may be a nearly constant ratio between the weight of the sample and that of the largest particle of ore contained therein.

In applying these principles to coal, slate and other impurities are to be considered as taking the place of the "richest mineral," but as coal is a low-priced commodity, the cost of collecting and preparing samples must necessarily be correspondingly small, making the installation and operation of elaborate sampling plants prohibitive.

Though the fineness to which the Bureau of Mines specifies that samples be crushed may not be strictly in accordance with the

[a] Brunton, D. W., The theory and practice of ore-sampling, Trans. Am. Inst. Min. Engrs., 1895, vol. 25, p. 827.

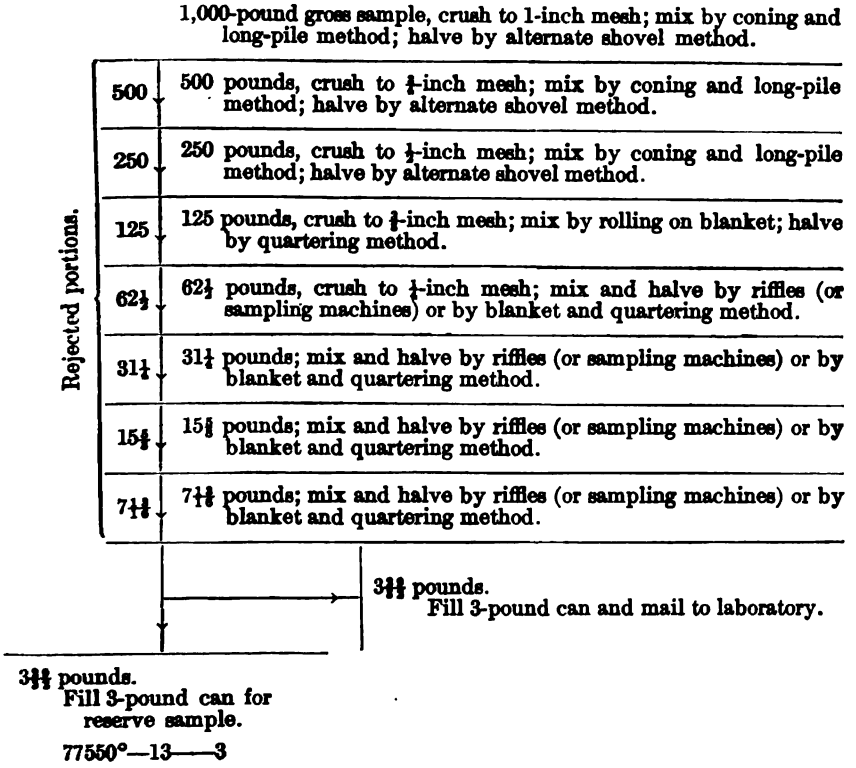
fineness that would be necessary for theoretically accurate reduction of the samples, and since, in fact, a coal of a definite physical character and composition would require special treatment, the fineness specified in the table below, it is believed, will give results that are fairly representative.

When prepared by hand, the pieces of coal and impurities should be crushed to the following approximate sizes before each reduction:

Weights of samples and corresponding sizes required.

Weight of sample to be divided.	Size to which coal and impurities should be broken before each division.
	Inch.
1,000 pounds or more.....	1
500 pounds.....	½
250 pounds.....	¼
125 pounds.....	⅛
60 pounds.....	¼

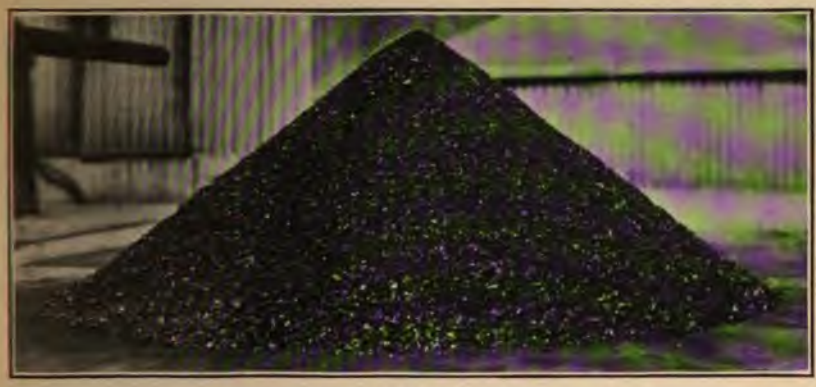
Diagram showing the treatment of a 1000-pound sample.



Care should be exercised to crush finely pieces of foreign matter before each reduction so that the crushed impurities can be distributed through the sample, and when crushing to keep pieces of slate and other impurities from flying out of the sample. In crushing the coal on wood, brick, cement, or floors of such materials splinters or small fragments may be broken from the floor and be mixed with the sample. Such floors, if used, should be thoroughly clean and free from cracks. If a sheet-iron plate is used, it should be free from rust.

After each crushing the sample should be thoroughly mixed before reduction in quantity. The method which gives generally satisfactory results is as follows:

The crushed coal is formed into a conical pile by depositing each shovelful of coal on top of the preceding one (Pl. I, *A*). (Pl. I, *A*, shows approximately 1,100 pounds of coal crushed to 1-inch mesh and finer, properly coned.) As the shovelfuls are deposited the fine material forms the apex of the cone, while the coarse particles roll down toward the base. By walking around the cone and systematically depositing shovelfuls on the apex of the cone from every side, care being taken to maintain the original form, the sampler will properly distribute the fine and coarse coal. A new, long, pile is then formed by taking a shovelful at a time (as the sampler fills the shovel he should walk around the cone, thus systematically removing the coal from the base of the cone) and spreading it out in a straight line or ribbon the width of the shovel; the length is 8 to 10 feet for a shovel holding about 15 pounds. Each new shovelful is spread over the top of the preceding one, beginning at opposite ends, and so on until all the coal has been formed into one long pile. (Pl. I, *B*, shows the coal of the cone formed into a long pile.) The sample is then halved in quantity by shoveling the long pile to one side, alternate shovelfuls being discarded while the retained shovelfuls are formed into a new cone. In shoveling the coal from the long pile the sampler takes the shovelfuls of coal systematically around the pile, advancing around at each shovelful a distance about equal to the width of the shovel, thereby preserving the symmetry in the form of the pile. If the pile should be reduced by shoveling all of the coal from either end, and if the alternate shovelfuls discarded contained coal mainly from the sides of the pile, the rejected half of the sample would contain a preponderance of coarse coal, while the retained half would contain relatively too much fine coal. This is because the coarse coal rolls down the sides of a pile or cone while the fine coal builds up, so that the relative proportions of coarse and fine coal in outer and inner portions of the pile are quite different. The alternate shovelfuls of coal which are retained are formed into a new cone.



A. CRUSHED GROSS SAMPLE IN CONICAL PILE.

B. CRUSHED GROSS SAMPLE REMIXED IN LONG PILE.

C. REDUCED SAMPLE QUARTERED.

In coning, care should be observed to deposit each shovelful so that the center of the cone as started will not be drawn to one side, for in quartering a cone the center of which has been drawn, two opposite quarters will contain an excess ratio of fine material, while the other two quarters will contain a deficiency. This will be apparent when it is considered that when a cone is formed, the fines build up the apex while the coarse particles roll down the sides. In ore sampling this may be of especial importance, since the fines are generally the richest ore, and as a result the metallic content of the final sample will be more or less than the average of the original pile, depending upon which two quarters are retained. D. W. Brunton, in a paper entitled "Modern Practice of Ore-Sampling,"[a] shows by illustrations and an example the effect of "drawing the center." He takes as an example the reduction of a 2,000-pound lot of ore to 62½ pounds (requiring quartering five times), and supposing that at each stage the sample taken represented 98 per cent of the actual value of the cone, he shows that the final sample would only give 90.3 per cent of the true value of the cone.

The above "coning," "long pile," and "alternate shovel" method of mixing and reducing the sample is followed until the sample is reduced to from 125 to 250 pounds. Samples smaller than 125 to 250 pounds are mixed on a canvas about 6 by 8 feet in dimension by raising first one end of the canvas and then the other, thereby rolling the sample back and forth. After being thoroughly mixed in this manner, the sample is formed into a conical pile by gathering the four corners of the blankets. The cone is then flattened, its apex being pressed down with the shovel or a board, so that each quarter contains the material originally in it. The flattened mass which should be of uniform thickness and diameter is then marked off into quarters with a board held edgewise, or a piece of sheet iron, along two lines that intersect at right angles directly under the apex of the original cone. The diagonally opposite quarters are shoveled away and discarded, and the spaces which they occupied brushed clean (Pl. I, *C*, shows the four quarters and the intersecting lines; quarters 1 and 3 will be rejected). The coal remaining is successively mixed, coned, and quartered on the canvas until two opposite quarters are equal to or somewhat less than the quantity (approximately 3 pounds) required to fill the sample can for shipment to the laboratory. If after two opposite quarters are placed in the sampling can, it is found that the can is not compactly filled, the other two quarters should be mixed, coned, flattened, and quartered, and the remaining space in the can then filled by taking equal segments from opposite quarters, using the sampling scoop (p. 29). The two rejected quarters are not thrown into the discard, but are placed in a sampling

[a] Am. Inst. Min. Engrs., vol. 40, p. 571, presented at the Spokane meeting, September, 1909.

can, hermetically sealed and held in reserve at the delivery point until report of analysis of the regular sample is received and settlement made for the coal.

The operations and methods followed in preparing and reducing the gross sample to a quantity convenient for transmittal to the laboratory are diagrammatically shown on page 33. The treatment of the 3-pound sample when it is received by the Bureau of Mines is diagrammatically shown on pages 40 and 41.

Accuracy in reducing the gross samples requires that the coal be crushed as directed, thoroughly and systematically mixed, formed in piles, and accurately divided, either by the alternate-shovel method or by quartering, so that the rejected portions and the retained portions will be uniform in character and weight. Thorough cleanliness must be maintained during the entire operation.

The method of coning and quartering coal and the principles which must be recognized are in essential features as given by Richards [a] for reducing ore samples.

Whenever the different increments of samples are collected throughout some considerable period of time, each increment may be crushed as soon as taken, and the pieces of coal and impurities may be broken sufficiently small to permit of two or more reductions of the accumulated samples before further crushing is necessary.

If deliveries extend over a considerable period, what would otherwise be a gross sample may be worked down, as it accumulates, in successive stages to quantities of a size suitable for transmittal to the laboratory, and the samples representing the several equal parts may be analyzed and the several analyses averaged, or they may later be mixed (at the delivery point or in the laboratory and reduced to one sample) and one analysis made.

If the contract amounts to a considerable quantity, necessitating several samples in relatively short periods, the installation of a crusher has been found in some cases expedient and economical for reducing the gross samples to one-quarter-inch mesh, or finer, and in such case, instead of the mixing and reducing being done by hand, the reducing buckets or mixing and reducing machine shown in Plate II are used. Even though the contracts are relatively small and the samples are crushed by hand, the use of the reducing bucket or machine is recommended. These devices are generally used after the gross sample has been mixed and has been reduced by the "alternate-shovel" or quartering method to about 60 pounds.

After the gross sample has been reduced to about 200 pounds, a shovel takes too large a proportion of the coal, making further reduction by alternate shovelfuls unreliable. The blanket and quartering methods are introduced after the sample has been reduced by

[a] Richards, R. H., Ore dressing, vol. 2, p. 844, and vol. 3, pp. 1574-1578.

alternate shovels to 125 to 250 pounds, as the capacity of the ordinary shovel is out of proportion to the size of the sample. It may prove desirable in sampling coals containing considerable quantities of free impurities of uneven size to use small shovels of 10 pounds and less capacity in reducing the gross sample by alternate shovels, for as the size of the shovel is diminished the number of fractional parts into which the sample is divided is increased. As the sample becomes smaller, greater precision in quartering is required, and, consequently, refinements in the manner of mixing and reducing should be introduced. The use of buckets or the machine makes thorough mixing possible and the riffles insure accurate division.

A mechanical crusher and mechanical means for preparing the samples for transmittal to the laboratory may be installed and the collection and preparation of the sample can then be made a continuous and purely mechanical process. When mechanical crushers are used the samples should be inspected before or while being fed to the crusher to make certain that the sample does not by chance contain a piece of iron, such as a tie spike, accidentally loaded with the coal in the mine and undetected in collecting the sample, or some other highly abrasive substance, otherwise the abrasion of the iron may introduce errors which would alter appreciably the final results.

SAMPLING BUCKETS.

When the sampling buckets are used, two similar ones are required. About half of the top of each bucket is covered with a riffle. The coal is placed in one of the buckets, and the projections *aa'* (Pl. II, *A*) of the stiffening rod of this bucket are placed in the notches *bb'* of the second bucket. When the coal is poured from the first bucket it flows to the riffle of the second bucket and every alternate section cut out by the riffle is discarded and the other sections are caught in the second bucket, thus halving the sample. The sample is poured from one bucket to the other until it is reduced to about 3 pounds, but the discarded 3-pound half is placed in a can and held as the reserve sample.

THE MIXING AND REDUCING MACHINE.

If more thorough mixing than can be done with the buckets is desired, the mixing and reducing machine is used. Plate II, *B*, shows the apron of the cylinder of this machine open for receiving a sample. After the sample is poured in, the apron's position is shifted, *a* being moved to *a'*. The cylinder is then closed and revolved counterclockwise. The closed sides of the riffles plow through and thoroughly mix the coal, and no coal can be discharged through the riffle while the cylinder is revolved in this direction if the level of the coal

is below the axis of the cylinder. After the sample has been mixed, the cylinder is rotated one turn clockwise; the coal in the cylinder is then cut by the planes of the riffle and half of it is discharged into the receiving tray. The coal remaining is again mixed by revolving the cylinder counterclockwise. By alternately changing the direction of rotation, the coal is alternately mixed and halved until about 6 pounds remain in the cylinder. The tray is then emptied of the discarded coal and the 6-pound sample mixed and halved, and the 3-pound sample caught in the tray is put in a can and becomes the official sample, while the 3-pound sample remaining in the cylinder is put in a can and held as the reserve sample. The interior of the machine is easy of access and should be brushed clean after each sample.

Generally speaking, the use of mechanical means for sampling coal and preparing samples gives more reliable and satisfactory results than hand labor, as the personal equation is partly eliminated.

COLLECTION OF SAMPLES IN THE DISTRICT OF COLUMBIA.

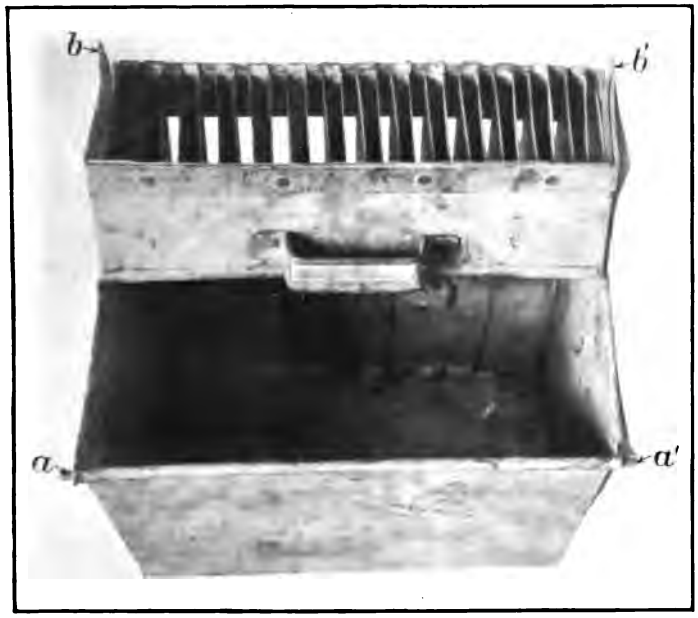
Gross samples are collected from deliveries under Government contracts in the District of Columbia in accordance with the method given in the specifications and as set forth in this paper. Instead of the gross sample being reduced to 3 pounds, as is necessary in the case of deliveries elsewhere, the sample after it is reduced to 125 to 250 pounds, is placed in galvanized-iron buckets, each large enough to hold about 70 pounds of coal and having a close-fitting lid that can be locked. The locked buckets of coal are immediately delivered to the crushing room in the Bureau of Mines building, where the preparation of the sample for the laboratory is continued.

THE CRUSHING ROOM.

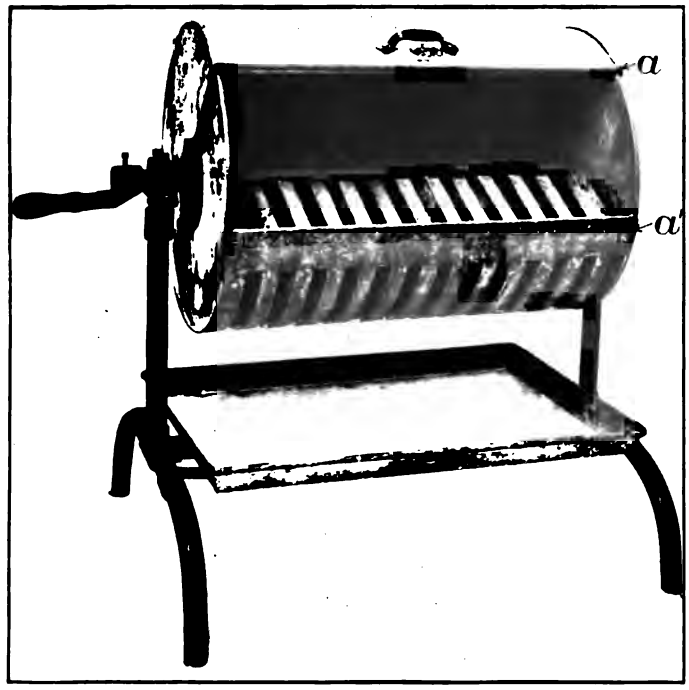
For preparing the samples collected in Washington, after they have been reduced at the delivery point to 125 to 250 pounds, specially designed machinery is installed in the crushing room of the Bureau of Mines plant. The samples are put through a motor-driven hammer crusher which crushes the coal to pass a one-quarter inch mesh.

Plate III is a view of the coal-crushing machinery through which the samples are passed. The crusher (not shown in the plate, being hidden by the extensions of the platform) rests on metal framework, its base being nearly on a level with the platform, which is 14 feet 4 inches from the floor of the crushing room. The buckets of coal are raised to the platform by means of a hydraulic elevator (not shown in the view). The samples are fed from the buckets to the crusher through the hopper.

After the entire sample has been crushed and caught in the bin or pocket (*a*, Pl. III) immediately below the crusher, it is passed into



A. SAMPLING BUCKET.



B. MIXING AND REDUCING MACHINE.

the revolving cylinder or mixer (*b*, Pl. III). This cylinder can be locked in position, with the discharge opening of the pocket registering with the opening in the mixer. When the sample is discharged from the pocket, the openings of the pocket and the cylinder are closed by suitable gates. By releasing the locking device and throwing in a clutch the cylinder is caused to revolve. The crusher and pocket are then free to receive another sample.

After the sample is thoroughly mixed, the mixer is stopped, its opening registered with the opening of the chute directly below the mixer, and the mixer locked in this position. The gate of the cylinder is then opened, and the coal is discharged into the separating chute (*c*, Pl. III).

Immediately below the opening at the top of the separating chute the coal passes over a riffle, which consists of a series of vertical parallel planes, equally spaced and so constructed that the coal when being passed over them is cut into sections. Every alternate section is deflected to the discharge, the other sections pass through and are retained. One-half of the coal is thus deflected into a receiving bin and eliminated from the sample. The other half falls vertically through the chute to a second riffle, where it is again halved, one half being carried to a second receiving bin, and the other half passing down to a third riffle. There are in all four riffles in the chute, and the quantity finally received in the receptacle directly below the chute is one-sixteenth of the quantity of the original sample that was received into the chute. By means of doors in the chute each riffle can be inspected and cleaned.

Four of the five bins for receiving discharged coal from the separating chute can be seen in Plate III (1, 2, 3, and 5). Four of these bins receive discarded coal, the largest bin receiving the discarded half from the first riffle, the next largest the discarded half from the second riffle, and so on, the final sample being received in the fifth bin (Pl. III, 5) directly below the chute.

Except for the platform, the crusher mechanism is constructed wholly of metal. The bins, chute, and other parts through which the coal is passed are made as dust proof as possible, iron cement being used to close leaks. However, more or less free dust arises from the crusher. This dust is carried away by means of a ventilating system. An exhaust fan driven by a direct-connected motor is located in the penthouse on the roof of the building. Galvanized-iron ducts lead from the fan to the laboratory, crushing room, and sampling room. In the crushing and sampling rooms, running off from the main ventilating duct (*d*, Pl. III), are branches connecting with each bin, some with hoods being placed over dust-producing machines. On each one of these branch ducts, as well as on the main

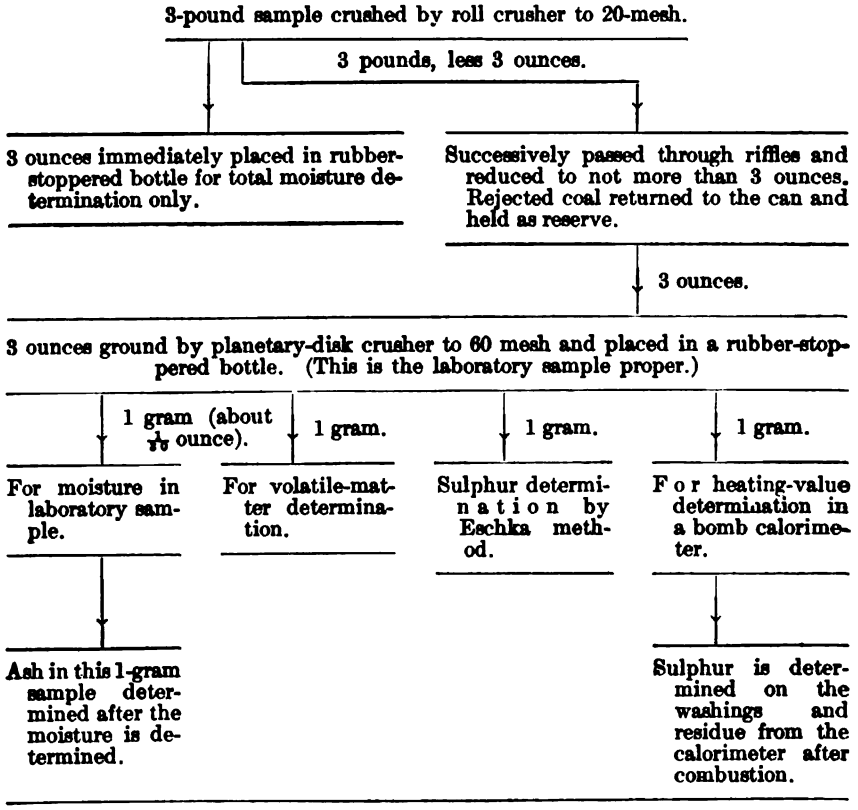
ventilating pipe, dampers are placed, by which the draft is regulated. The system works very satisfactorily and keeps the crushing and sampling rooms free from dust.

If the quantity of coal finally delivered from the chute is larger than is desired, it is further reduced by using two sampling buckets (Pl. II, *A*) or the reducing machine (Pl. II, *B*). The mixing machine is more generally used, because the machine mixes the coal more thoroughly than the buckets and prevents the sample from being exposed to the air.

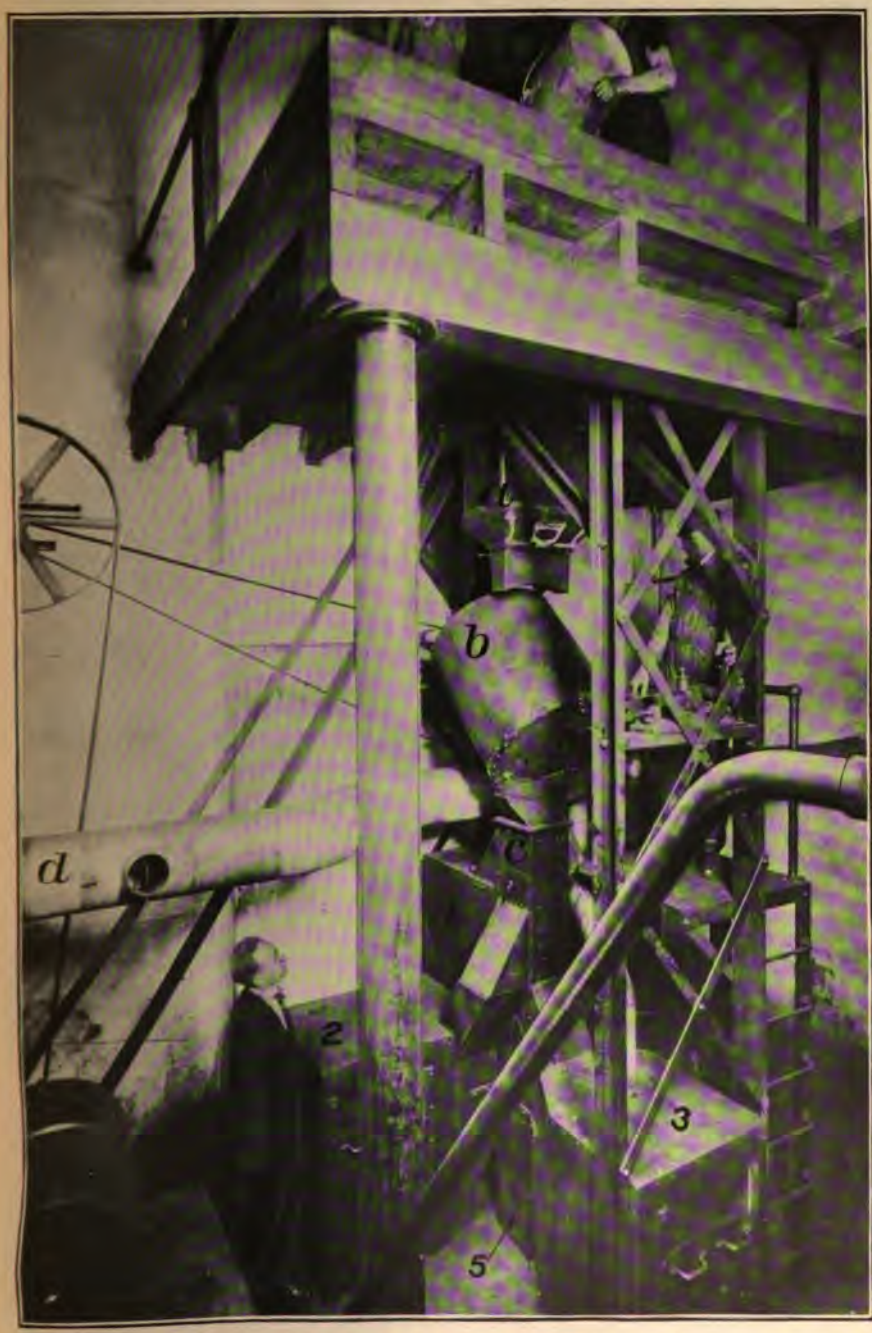
All samples are reduced to a quantity that will nearly fill a 3-pound galvanized can. The samples are then in the same stage of preparation as those received from points outside of Washington, and like them are further prepared in the sampling room.[a] The methods of preparation are diagrammatically shown below:

Diagrams showing treatment of the 3-pound sample in the laboratory.

METHOD A.



[a] The method of preparation of the samples in the sampling room, with a description of the laboratory methods and equipment, is given on p. 74 to 91, Bureau of Mines Bulletin 41.



MACHINERY FOR MIXING AND REDUCING GROSS SAMPLE.

The gross sample is accumulated in buckets, each holding approximately 70 pounds of coal, and the filled buckets are stored on the platform. Each bucket of coal making up the gross sample is successively passed through the motor-driven crusher which reduces the coal to ¼-inch mesh and finer (practically all of the coal will pass through ⅛-inch mesh), and the coal is halved by a riffle, one-half being

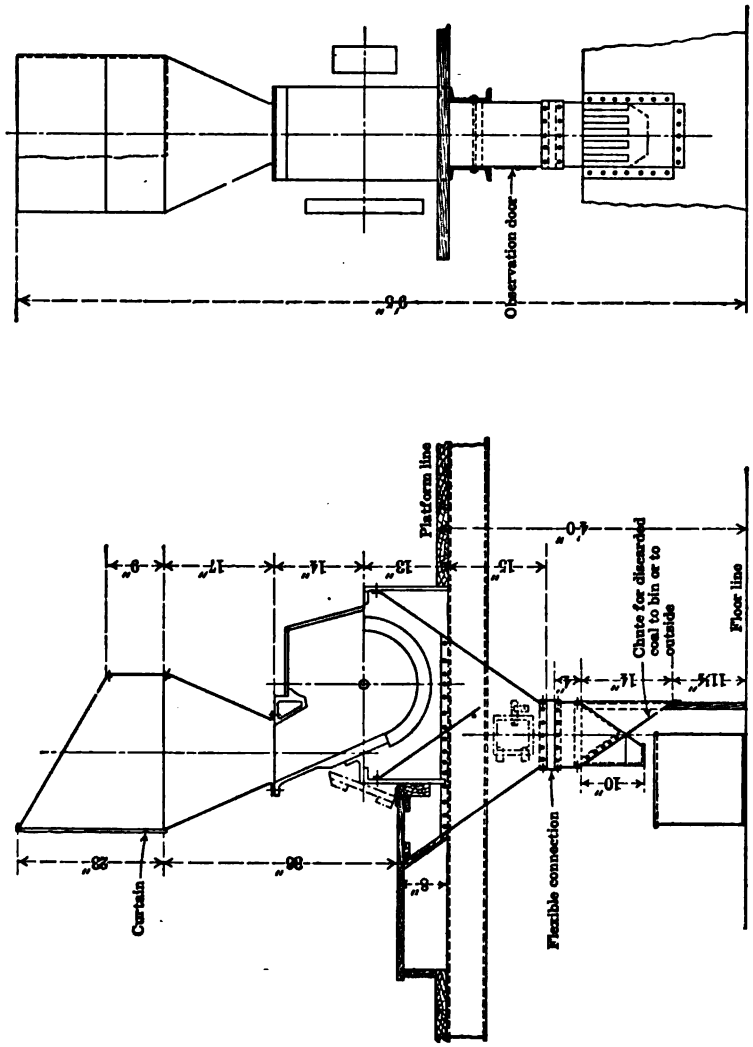


FIGURE 3.—Design of a simple coal-sampling plant.

deflected to a bin for discarded coal and the other half into a bucket. As the buckets are emptied into the crusher, they are available for catching the retained half from the riffle. Buckets containing the one-half of the gross sample are easily swung to the platform and the sample again halved in quantity by dumping the coal through the trapdoor (shown in fig. 3) and repassing it over the riffles. Elevating the sample to the platform and halving the sample are repeated

until the sample is reduced to the quantity convenient for transmittal to the laboratory.

Only the relation of one bin wall to the riffle is shown in the drawing; the shape of the bin can be varied to suit the arrangements in the building. If the building in which the installation is housed is so situated that a chute for the discarded coal can lead directly into a wagon body, car, or the boiler room, the disposal of this coal will be simplified.

For the inspection of the riffle and the removal of matter clogging the riffle a small door is provided in the main hopper. After each crushing the hopper and riffle should be brushed clean.

This particular form of equipment is developed with a view to simplicity and cheapness. By placing a series of riffles one below the other, the sample can be reduced to one-quarter, one-eighth, one-sixteenth, etc., of its original quantity, depending upon the number of riffles installed. As more riffles are added, the cost is increased and the platform must be raised, requiring other means than hand labor for elevating the parts of the samples. The Bureau of Mines is experimenting with other devices and types with a view to increasing the efficiency of the installation so that it will accurately select fractional parts of the gross sample with a minimum of handling and at a relatively low cost.

SEALING AND MAILING.

The final 3-pound sample is immediately placed in the galvanized-iron can furnished by the Bureau of Mines and sealed air-tight. The coal should be firmly packed in the can, so as to occupy as much of the space as possible, since in this way the air is more nearly excluded. This packing is best accomplished by having the coal finely crushed in the manner described in the preceding pages and by shaking or jarring the can repeatedly and vigorously while filling it.

The can is 11 inches in length and 3⅛ inches in diameter (inside dimensions), with a screw cap 2 inches in diameter. Its edges are crimped and carefully soldered so as to make it air-tight and strong. The screw cap has a gasket or washer of rubber or other flexible material to exclude the air. As a further protection and to insure tightness the cap when in place and screwed down is wrapped carefully with several layers of adhesive tape, the first layer of which completely covers the joint between the lower edge of the cap and the neck of the can. In Plate IV, *a* shows the first layer of the tape being forced down with thumb and forefinger; *b* the can properly sealed, and *c* the can ready for mailing. It is not advisable to use solder, paraffin, or sealing wax of any kind, because some of the material may be mixed with the coal, either when it is applied or when the cap is removed. Before being filled each can should be carefully inspected as to tightness and freedom from rust.

After the can is properly sealed data concerning the sample are properly recorded on a blank form furnished by the Bureau of

Mines and the form is placed (not pasted) around the can, an addressed franked wrapper (*c*, Pl. IV), is put over it and the can is mailed. The wrapper has a gummed end, but to insure its sticking and keeping the form from being lost a rubber band is put on or a string is tightly tied around the can, as shown in the illustration.

The blank form is in two sections—an original and a duplicate. The original form accompanies the sample, furnishes the bureau complete information as to the sample, and becomes the bureau's permanent record. The duplicate is used as a file copy for the office submitting the sample.

Special attention should be given to filling out the form and promptly forwarding the sample. Failure to fill out the form properly or to place it around the can may cause indefinite delay in making the analysis of a sample and in reporting the results.

A letter should be written informing the bureau that a sample or samples (giving can numbers) have been forwarded. As the tonnage on many orders or deliveries during a month may be considerable, and as samples may be submitted from time to time, unless the bureau is informed it has no knowledge whether all samples have been received to represent the tonnage, nor has it any intimation as to which is the first or last sample, although this information is necessary to insure prompt reports. To furnish the necessary information, it should be stated in the letter, as follows:

This is the first and only sample which will be submitted on order No. ——, or of the deliveries during the month of ———.

Or,

This sample is the —— (give number) one submitted on order No. ——, or of the deliveries during the month of ———, and other samples on this order or month's delivery will be submitted.

Or,

This is the last sample on order No. ——, or of the deliveries during the month of ———, and is to be considered with samples submitted in cans numbered ———.

If a sample is submitted representing coal from a mine or from a source other than that specified in the contract, the letter transmitting the sample should so state, in order that the records of the Bureau of Mines will be complete.

A copy of the front and reverse sides of the two sections comprising the form mentioned above are herewith given.

Forms for reporting samples.

[Original, front.]

DEPARTMENT OF THE INTERIOR.

BUREAU OF MINES.

Information to accompany each sample of coal submitted for analysis.

Coal delivered to .....

(If ship or barge, state where loaded.)

Number of can ..... Number of sample .....

Name of contractor .....

SEALING AND LABELING SAMPLE CAN.

Coal, kind and size .....
Number of tons or pounds delivered represented by this sample .....
Dates of delivery .....
Number of contract ..... Number of order .....
Date of mailing sample ..... Date received .....
For the bureau's information in reporting on an order or month's delivery, state whether this sample is the first, last, or only sample, or whether it is to be considered with other samples on this order or month's delivery .....
.....
.....

Original. Signed.....

[Original, back.]

SUGGESTIONS AS TO FILLING AND SEALING CANS.

The coal sample should be firmly packed in the can, so as to occupy as much of the space as possible, since in this way the air is more nearly excluded. To insure the tightness of the can, the cap when in place and screwed down should be wrapped carefully with several layers of adhesive tape, the first layer of which should completely cover the joint between the lower edge of the cap and the neck of the can. It is not advisable to use solder, paraffin, or sealing wax of any kind, because some of the material may be mixed with the coal either when the material is applied or when the cap is removed. Before being filled each can should be carefully inspected as to tightness and freedom from rust. The Bureau of Mines will furnish a month's supply of cans, addressed postal-frank mailing wrappers, and this form; if additional supplies are needed they can be obtained from the bureau monthly on request.

[For instructions as to use of this form, see instructions on back of attached (duplicate) section.]

[Duplicate, front.]

DEPARTMENT OF THE INTERIOR.

BUREAU OF MINES.

Coal delivered to.....
(If ship or barge, state where loaded.)
Number of can..... Number of sample.....
Name of contractor.....
Coal, kind and size.....
Number of tons or pounds delivered represented by this sample.....
Dates of delivery.....
Number of contract..... Number of order.....
Date of mailing sample..... Date received.....
For the bureau's information in reporting on an order or month's delivery, state whether this sample is the first, last, or only sample, or whether it is to be considered with other samples on this order or month's delivery.....
.....
.....

Duplicate. Signed.....

[Duplicate, back.]

INSTRUCTIONS AS TO THE USE OF THIS FORM.

Fill out this form for each sample of coal forwarded to the Bureau of Mines; the office filling out the form is to retain the *duplicate* copy. The original must be placed around the can (but not pasted to the can—it may be held in place by a rubber band), under the addressed postal-frank wrapper, in such a manner that the sample and the information regarding it can not be separated in mailing. When the sample reaches the Bureau of Mines the franked wrapper is torn off and complete information relative

to the sample is found on the card. This form must be properly filled out and forwarded in order to expedite testing and reporting. A letter should be written informing the Bureau that sample or samples (giving can numbers) have been forwarded. The receipt of the sample will be immediately acknowledged.

[For suggestions as to filling and sealing cans, see back of attached (original) section.]

REPORTING ANALYSES

The analyses and tests of samples are reported by the laboratory to the fuel-inspection section, where the record of the samples is kept. The laboratory receives the samples with an identification number only, which is given the sample by the fuel-inspection section when the sample is received from the mail, and the laboratory reports the analysis and test by number only. If a report is to be based on the analyses of two or more samples, the analyses are averaged, each analysis being given a weight according to the proportion of the total tonnage which it represented. The results are then reported in triplicate on the following letter form to the branch of the service submitting the sample or samples, which in turn furnishes the contractor one of the copies:

Form for reporting analysis.

Serial No. .....

DEPARTMENT OF THE INTERIOR.

BUREAU OF MINES.

Washington, D. C., ...

SIR: In reference to the ..... tons of ..... coal delivered on your contract No. ..... and your order No. ..... to the ...., 191 , by.....

The analysis of the samples of coal received by the Bureau of Mines was as follows:

	As received.	Dry coal.
Moisture.....		
Volatile matter.....		
Fixed carbon.....		
Ash.....		
Total.....		
Sulphur.....		
British thermal units.....		

This information is for the use of the Government and the dealer or operator furnishing the coal. It is confidential until it is published by the United States Government.

Remarks:

Certified:

.....
Engineer in charge
Fuel Inspection.

Respectfully,

.....
Assistant to the Director.

COAL SPECIFICATIONS AND PROPOSALS.

There are given in the following pages coal specifications and proposals typical of those in use by the Government for the purchase of coal under specifications based on the heating value, the content of ash and of moisture, and other considerations.

Two classifications of the coal purchased are made, and specifications accordingly prepared, namely, (*a*) coals for steam power plants, which include bituminous (semibituminous, subbituminous, lignite) and the small sizes of anthracite (pea and buckwheat No. 1), and (*b*) the large sizes of anthracite (broken, egg, chestnut, and stove), which are used in small house-heating furnaces and stoves for heating and domestic purposes. The specifications for steam power plants can be readily modified to provide for the purchase of smaller sizes (see table, p. 51) of anthracite—that is, coal which will pass through a mesh one-fourth inch square. If it is desired to purchase any particular coal, a specification can readily be prepared from the two types of specifications herein presented.

The purpose of the specifications is to clearly set forth the character and quality of coal desired, to obtain bids which fully specify the coal offered, the values offered in the case of bituminous coal by the successful bidder becoming a part of the contract, to furnish means whereby the Government may be assured of receiving the coal contracted for, providing for a definite procedure for determining equitable settlement for coals differing in character and quality, and to include concise statements of the necessary legal phases, etc., all with the view that the specifications and proposal signed by the successful bidder may become a contract which will work to the advantage and justice of both the Government and the contractor.

Attention is especially directed to instructions for sampling given in the specifications and which become a part of the contract. Prior to the fiscal year 1912–13 the method of sampling was given scant mention, leaving one of the most important features determining the successful and satisfactory application of the specification method of the purchase of coal almost wholly in the hands of the purchaser, the seller having little or no means whereby he could exercise a control of the manner and method of collecting and preparing samples that should represent the coal furnished. It is as essential that an established and an agreed-upon method of sampling be followed, as it is that the sample be analyzed by a reputable laboratory; in fact, the sampling is of first consequence, for if a sample is haphazardly taken and carelessly reduced to a quantity convenient for transmittal to the laboratory, and hence is nonrepresentative, it is usually impossible or impracticable to get another sample; whereas if the analysis and test of a representative sample are in error, the remaining part of the sample can be analyzed and tested, and by another

laboratory if desired. Unwarranted corrections in price and resulting injustice and hardship to the buyer or seller may easily and unintentionally occur through lack of perception of the relative importance of sampling and the observance of proper methods of collecting and preparing samples. By clearly setting forth the method of sampling as a part of the contract, the seller or the buyer has something tangible on which to base legitimate complaints if samples are not properly collected and prepared. Such is the purpose of putting in the Government specifications the method of sampling in some detail.

As the specifications given herein are so general, applying generally to all coals purchased by the Government under every condition of delivery, it is impossible to state therein specific directions for sampling, but for any particular conditions of delivery more definite directions can be prepared and proper consideration for the character of the coal given. By observance, however, of the general directions for sampling, which are given in the specifications that follow, samples that are fairly representative of the coal sampled can be obtained without difficulty.

Though a contract may be let under the specification basis, the sampling of small orders of, say, 25 tons or less, may be discretionary with the Government, a provision to this effect appearing in the contract. If samples are not taken the bid price per ton is paid, unless coal is delivered which gives unsatisfactory results because of excessive clinkering, excessive ash, or any other cause which would make it subject to rejection, in which case the procedure under section 48, "Unsatisfactory fuel," of the bituminous specifications would be followed. This discretionary right of sampling small orders was included in the specifications prepared in the office of the chief of the Quartermaster Corps for the purchase of coal for the Army stations in the United States for the fiscal year 1912–13. This provision made it unnecessary to sample deliveries under one large general contract of from a half to a few tons delivered to small launches or boats, or the different quarters where facilities for sampling or storing accumulated samples were not at hand. It is obvious that the expense of collecting and preparing a sample to represent such small tonnages, together with the cost of analysis, would make the unit cost per ton ordinarily prohibitive and disadvantageous.

Under "Price and Payment" for bituminous coal, page 60, 2 per cent variation from the guaranteed standard of British thermal units (B. t. u.) is allowed before price corrections. On 14,000 B. t. u., "dry coal," this means that there is no price correction for delivered coal of an apparent heating value as reported by the laboratory between 13,720 and 14,280. This is allowed to provide (*a*) for the reasonable variations that, it is recognized, may be obtained with the

same 3-pound sample in two different determinations in the same laboratory, (*b*) for variations that result in preparing and reducing the gross sample to a quantity convenient for transmittal to the laboratory, (*c*) for variations due to the collection of gross samples as representing the coal sampled, and (*d*) for allowing the contractor latitude in the preparation of the coal, since it is recognized that the quality of his coal expressed in terms of ash and B. t. u. can not be controlled within strictly narrow limits.

In interpreting a heating value determination, then, the value reported by the laboratory must be considered as the apparent value of the coal sampled, as the actual value of the coal may have been more or less than the apparent value because of the variations above cited. If the reported value differs by more than 2 per cent from the guaranteed standard, the result is considered as evidence that coal of a quality other than that contracted for was delivered and the price is corrected accordingly. If a number of gross samples are taken to represent a total quantity of coal delivered, and a report is obtained by averaging the several analyses and tests, the apparent value reported by the laboratory may be within 1 per cent or less of the actual value of the entire quantity, but in specifications as general as those herein given a 2 per cent variation from the standard is considered more equitable and will eliminate frequent and unwarranted price corrections, especially if payment for a delivery or order is based on a single determination. In the sampling of cargo shipments to the Isthmus of Panama (p. 25), the reported results between two separate and independent series of samples representing the same cargo agree on an average within less than 50 B. t. u. The samples are collected by experienced collectors and prepared very systematically, and the reported results for a cargo are determined by averaging a number of analyses. With such sampling the variations above enumerated are largely averaged and eliminated, and the variation that may be allowed before price corrections may, under such conditions, be reduced to 1 per cent or even less.

It is not the intent of the specifications continually to make corrections in price for slight variations of the actual heating value from the guaranteed standard, and it is desired to eliminate corrections which are unwarranted because of the difference between the reported apparent heating value and the actual heating value, the first purpose of the specifications being to insure the Government is receiving coal similar within reasonable limits to the standard of the contract.

The bituminous specifications require the bidder to specify the quality of coal offered in terms of moisture, "as received," ash, sulphur, volatile matter, and B. t. u., "dry coal;" these values becoming the standards of the contract to determine rejectable coal and the price to be paid for delivered coal. Bidders have in some cases

specified a higher B. t. u. value and a lower ash content than the quality of the coal warranted, upon the theory that the high B. t. u. and low ash values would give them an advantage in the comparison of bids. The bid price may be correspondingly higher, the bidder presumably expecting a deduction to be made and to receive a price on his delivered coal lower than the bid price. The apparent price may be, therefore, higher than the actual price received, and use may be made of the apparent price for advertising purposes. Bidders are cautioned against offering higher standards than can be maintained on an average, for to do so may result in the bid being rejected (see sec. 8, p. 53, of the specifications) or may result in excessive penalties, the purchase of the coal in the open market, the difference in cost of coal so purchased being charged against the contractor, or may result in the cancellation of the contract (see sec. 48, p. 59).

By reference to the specifications for anthracite, broken, egg, stove, and chestnut sizes, it will be noted that bidders are not required to specify the quality of the coal in terms of ash, B. t. u., moisture, sulphur, and volatile matter, and that regular and systematic sampling of all deliveries is not required to determine the price to be paid for each and every delivery. The specifications provide, however, for the sampling of coal that proves unsatisfactory because of excessive ash or clinkering and for price corrections if the ash as shown by analysis is in excess of a permitted variation from the standard for the particular size of coal, these standards being determined from hundreds of ash determinations made during a period of six years. B. t. u. standards are not used, as it has been determined that the heating value for the different sizes is, in the main, a function of the ash content, varying inversely and in proportion with the ash; that is to say, if the ash content is increased, for example, 1 per cent, the heating value is decreased 1 per cent.

Because of economic considerations and the physical character of anthracite, the specifications used for bituminous coal requiring sampling are not adapted for the purchase of the large sizes of anthracite. Most of the Government contracts for these sizes are for relatively small yearly tonnages with deliveries of ½ ton to 50 tons, but seldom more to a building. To collect regularly samples from such small deliveries would make the unit cost per ton of sampling prohibitive, the cost being generally greater than any probable saving. To obtain samples that would fairly represent the quality of these coals would require collecting samples of 1,000 to 1,500 pounds, or more, and to obtain a final sample for analysis that would represent the gross sample within reasonable limits would require crushing and recrushing the gross sample, thereby perhaps destroying the value of the the gross sample for use in a house-heating furnace or stove, and largely wasting it, unless a Government steam-power plant were

available in which it could be advantageously burned. The total or partial loss in value of 1,000 or 1,500 pounds of coal at $5 to $10 per ton from a small delivery is manifestly a matter deserving consideration. Owing to the hardness and brittleness of anthracite, the crushing of a gross sample by hand is difficult and tedious and requires hours of labor; in fact, a mechanical means of crushing is almost absolutely necessary, but the cost of the necessary equipment would prove prohibitive in the case of small contracts.

Because of the above facts, as well as the manner in which anthracite is prepared at the colliery and the small area of the Pennsylvania anthracite fields, and because of other considerations, the Government does not consider the application of the specifications which are adapted for bituminous coal generally suitable for the purchase of the large sizes of anthracite. The specifications that are used provide for the acceptance and use of coal and payment at contract price so long as no difficulty is experienced, and the collection of samples is necessary only as set forth in the specifications.

Anthracite is graded into sizes before shipment, and each grade commands a certain price. The size of the openings in screens and the types of screens used in preparing the sizes may vary slightly; however, the following table indicates the commercial sizes of anthracite:

Sizes of anthracite

[Prepared with screens of square mesh.]

Sizes.	Through meshes having a diameter of—	Over meshes having a diameter of—
	Inches.	*Inches.*
Broken (furnace)	4	2¾
Egg	2¾	2
Stove	2	1¼
Chestnut	1¼	¾
Pea	¾	½
Buckwheat No. 1	½	¼
Buckwheat No. 2	¼	9/32
Buckwheat No. 3	9/32	5/32

SPECIFICATIONS AND PROPOSALS FOR BITUMINOUS AND ANTHRACITE PEA AND BUCKWHEAT NO. 1[a] COAL FOR STEAM POWER PLANTS.

I. PROPOSALS.

1. Sealed proposals, in duplicate, to furnish the quantities of coal specified in the schedule herewith, required for the use of (the name of department, bureau, or institution) for the fiscal year ending June 30, 19.., will be received until 2'o'clock p. m., ..., 19.., at the office of....., and then opened.

2. Each bidder shall have the right to be present either in person or by attorney when the bids are opened.

II. ADDRESS OF PROPOSALS.

3. Proposals, in duplicate, must be forwarded to..... ....., postage prepaid. Addressed envelope for mailing is inclosed herewith.

III. PROPOSALS—GUARANTY.

Signature.

4. Proposals must be made in duplicate on the form furnished by ....., and must be signed by the individual, partnership, or corporation making the same; when made by a partnership, the name of each partner must be signed. If made by a corporation, proposals must be signed by the officer thereof authorized to bind it by contract and be accompanied by a copy, under seal, of his authority to sign.

Cash or certified check.

5. The proposals must be accompanied by cash or by a certified check drawn payable to the order of the Secretary of the ...., in an amount equal to 2 per cent of the estimated cost of the items for which bids are submitted, the minimum amount in any case to be $10. This requirement is solely to guarantee, if an award is made on the proposal, that within 10 days after notice is given that an award has been made, the bidder will enter into a contract in accordance with the terms of the proposal and execute a bond for the faithful performance thereof, with good and sufficient sureties as hereinafter required. In the event of the failure of the bidder to enter into contract or execute bond, the cash or check guaranty will be forfeited.

IV. CONTRACTOR'S BOND.

Sureties.

6. Each contractor shall be required to give a bond, with two or more individual sureties or one corporate surety duly qualified under the act of Congress approved August 13, 1894, in which they shall

[a] If smaller sizes of anthracite are required, the specifications can readily be modified to include them.

covenant and agree that, in case the said contractor shall fail to do or perform any or all of the covenants, stipulations, and agreements of said contract on the part of the said contractor to be performed as therein set forth, the said contractor and his sureties shall forfeit and pay to the United States of America any and all damages sustained by the United States by reason of any failure of the contractor fully and faithfully to keep and perform the terms and conditions of his contract to be recovered in an action at law in the name of the United States in any proper court of competent jurisdiction. Such sureties (except corporate sureties) shall justify their responsibility by affidavit showing that they severally own and possess property of the clear value in the aggregate of double the amount of the above-mentioned forfeiture over and above all debts and liabilities and all property by law exempt from execution; the affidavit shall be sworn to before a judge or a clerk of a court of record or a United States attorney, who must certify of his own personal knowledge that the sureties are sufficient to pay the full penalty of the bond.

May be waived.

7. If the estimated amount involved in the contract does not exceed the sum of $200, then the bond may be waived with the consent of the department involved.

V. RESERVATIONS.

Rejection and annulment.

8. The right will be reserved by the Secretary of ..... to reject any and all bids, to waive technical defects, and to accept any part of any bid and reject the other part, if, in his judgment, the interests of the Government shall so require; also the right to annul any contract, if, in his opinion, there shall be a failure at any time to perform faithfully any of its stipulations, or in case of a willful attempt to impose upon the Government coal inferior to that required by the contract; and any action taken in pursuance of this latter stipulation shall not affect or impair any right or claim of the United States to damages for the breach of any of the covenants of the contract by the contractor. Bidders are cautioned against guaranteeing higher standards of quality than can be maintained in delivered coal (this applies more especially to bituminous coal), as the Government reserves the right to reject any and all bids if the analyses and test results which the Government may have on record indicate that higher standards have been offered than can probably be maintained.

Estimated quantity.

9. The estimated quantity of coal to be purchased will be based upon the previous annual consumption, but the right will be reserved to order a greater or less quantity, subject to the actual requirements of the service.

Tests.

10. The right will be reserved by the Government to purchase, for the purpose of making boiler tests, other coal than that herein contracted for, provided the amount so purchased does not exceed five per cent (5 per cent) of the coal used at the plant during the period covered by this agreement.

Lowest bids may not be considered.

11. If it should appear to the best interests of the Government to do so, the right is reserved to award the contract for supplying coal at a price higher than that named in a lower bid or in lower bids, on the basis of the quality of the coal offered.

Failure to contract.

12. If the bidder to whom the award is made should fail to enter into a contract as herein provided, then the award may be annulled and the

contract let to the next most desirable bidder without further advertisement, and such bidder shall be required to fulfill every stipulation expressed herein, as if he were the original party to whom the contract was awarded.

Contracts nontransferable.

13. No contract can be lawfully transferred or assigned.

Default.

14. No proposal will be considered from any person, firm, or corporation in default of the performance of any contract or agreement made with the United States, or conclusively shown to have failed to perform satisfactorily such contract or agreement.

VI. DESCRIPTION OF COALS DESIRED.

(A) BITUMINOUS.

Coal desired.

15. Bids are desired on coal as specified below:

The coal must be a good steam {coking / noncoking} {run-of-mine / slack / lump (give size)}, bituminous coal, free from bone, slate, dirt, and excessive dust, and adapted for successful use in the particular furnace equipment.

Lowest quality acceptable.

16. Bituminous coal that shows on analysis a quality lower than that indicated below will not be considered:[a]

Moisture in "delivered coal"..... .... per cent.
Ash in "dry coal"..... .... per cent.
Volatile matter in "dry coal"..... .... per cent.
Sulphur (separately determined) in "dry coal"..... .... per cent.
British thermal units in "dry coal" below..... .....

Information to be supplied by bidder.

17. Bidders are required to specify the coal offered in terms of moisture, "as received"; ash, volatile matter, sulphur, and British thermal units, "dry coal"; which values become the standards for the coal of the successful bidder. In addition, the bidders are required to give the name and location of the mine producing the coal, the name or other designation of the coal bed, name of operator of mine, and the trade name of the coal. This information to be furnished in spaces provided under section 57 (A).

(B) ANTHRACITE PEA AND BUCKWHEAT NO. I.

Character.

18. The coal must be best quality Pennsylvania white-ash anthracite.

Quality.

19. It must be well screened, practically free from dirt, must not contain undue percentages of moisture, slate, or bone, and must equal in quality and preparation the best white-ash anthracite coal produced. The coal must not contain more than the following percentages of ash on the "dry-coal" basis:

Size of coal.	*Ash, "dry coal."*
Pea.....	16 per cent.
Buckwheat No. 1.....	18 per cent.

Information to be supplied by bidder.

20. The bidder will be required to specify the coal field, district, and colliery producing the coal, the name and address of the operator, and the railroad upon which the colliery is located. This information to be given in spaces provided under section 57 (B).

[a] The percentages and the heating value to be filled in by the office inviting bids.

VII. AWARD.

(A) BITUMINOUS.

Considerations.

21. Bids will be considered on each item separately; and in determining the award of the contract consideration will be given to the quality of the coal (expressed in terms of ash in "dry coal," of moisture in coal "as received," and of British thermal units in "dry coal") offered by the respective bidders, to the operating results obtained with the coals on previous Government contracts, as well as to the price per ton.

Method of comparing bids.

22. In order to compare bids as to the quality of the coal offered, all proposals will be adjusted to a common basis. The method used will be to merge all four variables—ash, moisture, calorific value, and price bid per ton—into one figure, the cost of 1,000,000 British thermal units, so that one bid may readily be compared with another. The procedure under this method will be as follows:

(*a*) All bids will be adjusted to the same ash percentage by selecting as the standard the proposal that offers coal containing the highest percentage of ash. Each 1 per cent of ash content below that of this standard will be assumed to have a positive value of 2 cents per ton, and the price will be accordingly decreased 2 cents, the amount of premium that is allowed under the contract for 1 per cent less ash than the standard established in the contract. Fractions of a per cent will be given proportional values. The adjusted bids will be figured to the nearest tenth of a cent.

(*b*) To reduce bids to a common basis with respect to the moisture in the coal offered, the price quoted, adjusted in accordance with the above, will be divided by the difference between 100 per cent and the per cent of moisture guaranteed in the proposal. The adjusted bids will be figured to the nearest tenth of a cent.

(*c*) On the basis of the adjusted price, allowance will then be made for the varying heat values by computing the cost of 1,000,000 British thermal units for each coal offered. This determination will be made by multiplying the adjusted price per ton by 1,000,000 and dividing the result by the product of 2,240, multiplied by the number of British thermal units guaranteed.

Service results and test.

23. If from practical service experience or by test any of the coals have in the past proved unsuited for the furnace and boiler equipment, or have failed to meet the requirements of the city smoke ordinances, the bids thereon may be eliminated from further consideration, regardless of their calculated costs per million B. t. u. The selection of the lowest bid of the remaining bids on the basis of the cost per million B. t. u. will be considered as a tentative award only, the Government reserving the right to have practical service test or tests made under the direction of the Bureau of Mines, the results to determine the final award of contract. The interested bidder or his authorized representatives may be present at such test.

(B) ANTHRACITE PEA, AND BUCKWHEAT NO. 1.

Considerations.

24. Bids will be considered on each item separately; and in determining the award of the contract, consideration will be given to the operating results obtained with similar coals on previous Government contracts offered by the respective bidders, as well as to the price per ton.

Service test.

25. Before making final award of contract, practical service tests, under the direction of the Bureau of Mines, will be made of selected coals to determine the suitability and adaptability of the coals for the particular furnace and boiler equipment concerend. The interested bidders or their authorized representatives may be present at such tests. Samples will be collected from the coals tested, and complete analyses and calorimetric determinations will be made, as well as determinations of the fusing temperature of the coal ash.

Test results part of contract.

26. The results of the service test of the coal that proves a satisfactory fuel and the results of the analyses and tests of the samples of that coal will become a part of the contract establishing the standard of the coal offered by the bidder to whom the final award is made. The results become the basis for determining rejectable coal during the life of the contract.

VIII. DELIVERY.

Quantity.

27. The coal shall be delivered in such quantities at such times as the Government may direct.

Rapidity.

28. All the available storage capacity of the Government coal bunkers will be placed at the disposal of the contractor to facilitate delivery of coal under favorable conditions. When an order is issued for coal, the contractor upon commencing a delivery on that order shall continue the delivery with such rapidity as not to waste unduly the services of the Government inspector. Information is furnished in the schedule herewith in relation to the several places, etc., for the delivery of coal, but the bidder is invited to visit those places and inspect the conditions for his own information.

Notices.

29. After verbal or written notice has been given to deliver coal under this contract, a second notice may be served in writing upon the contractor to make delivery of the coal so ordered within 24 hours after receipt of said second notice. Should the contractor, for any reason, fail to comply with the second request, the Government will be at liberty to buy coal in the open market and for coal so purchased to charge against the contractor and his sureties any excess in price over the contract price.

Hours.

30. The contractor will be allowed to deliver coal during the usual hours of teaming, that is, between 8 a. m. and 5 p. m.

Weighing.

31. The coal will be weighed by representatives of the Government without expense to the contractor.

IX. SAMPLING.

Imperative for bituminous coal.

32. As payment for bituminous coal is based upon the quality of coal delivered as shown by analyses of representative samples, it is imperative that samples representing every order of coal be taken and that the proper officials of the Government buildings see that such samples are obtained. The Bureau of Mines will have general direction of sampling, giving instructions in the methods of obtaining representative samples, and lending all practicable assistance.

Contractor may be present.

33. If desired by the coal contractor, permission will be given to him or his representative to be present and witness the collection and preparation of the samples to be forwarded to the Government laboratories.

During unloading.

34. The coal will be sampled at the time it is being loaded or unloaded from railroad cars, ships, barges, or wagons, or when discharged

from contractors' supply bins, except as provided for in section 27 of the specifications for anthracite broken, egg, stove, and chestnut.

Size of increments.

35. When the coal is being unloaded, a shovel or specially designed tool will be used for taking portions or increments of 10 to 30 pounds of coal. For slack or small sizes of anthracite increments as small as 5 to 10 pounds may be taken. The size of the increments depends on the size and weight of the largest pieces of coal and impurities.

Collection of gross sample.

36. The increments will be regularly and systematically collected, so that the entire delivery will be represented proportionately in the gross sample. The frequency of collecting the increments should be regulated so that a gross sample of not less than 1,000 pounds will be collected. If the coal contains an unusual amount of impurities, such as slate, and if the pieces of such impurities are very large, it will be necessary to collect gross samples of even 1,500 pounds or more. For slack coal and for small sizes of anthracite, if the impurities do not exist in abnormal quantities or in pieces larger than three-quarters of an inch, a gross sample of approximately 600 pounds may prove sufficient. The gross sample should contain the same proportion of lump coal, fine coal, and impurities as is contained in the coal delivered. As the increments are collected, they will be deposited in a receptacle having a tight-fitting lid and provided with a lock.

Preparation of gross sample.

37. After the gross sample is collected, it will be systematically crushed, mixed, and reduced in quantity to convenient size for transmittal to the laboratory. The crushing will be done by a mechanical crusher or by hand with an iron tamping bar on a smooth and solid floor. In the absence of a smooth, tight floor, the crushing may be done on a heavy canvas, to prevent the accidental admixture of any foreign matter. The mixing and reduction will be done by hand with a shovel, or mechanically by means of riffles or sampling machines.

Hand preparation.

38. When prepared by hand, the pieces of coal and impurities will be crushed to the approximate size indicated in the table below before each reduction:

Weight of sample to be divided.	Size to which coal and impurities should be broken before each division.
1,000 pounds or more..	1 inch.
500 pounds.....	$\frac{3}{4}$ inch.
250 pounds.....	$\frac{1}{2}$ inch.
125 pounds.....	$\frac{3}{8}$ inch.
60 pounds.....	$\frac{1}{4}$ inch.

The 60-pound sample will then be reduced by quartering, or by the use of riffles or sampling machines, to the desired quantity for transmittal to the laboratory.

Mixing and reduction by "alternate shovelfuls."

39. After each crushing, the sample will be thoroughly mixed before reduction in quantity, the procedure being as follows:

The crushed coal will be shoveled into a conical pile. A new long pile will then be formed by taking a shovelful at a time and spreading it out in a straight line (8 to 10 feet long for a shovel holding 15 pounds). Each new shovelful will be spread over the top of the preceding one,

beginning at opposite ends, and so on, until all the coal has been formed into one long pile. By walking around the long pile and systematically taking shovelfuls, and shoveling the coal to one side, alternate shovelfuls being discarded, the sample will be halved in quantity.

Mixing and reduction by "quartering."

40. The above "long-pile" and "alternate-shovel" method of mixing and reducing the sample will be followed with samples of 125 to 250 pounds or more. Samples smaller will be mixed on a canvas about 8 feet square by raising first one end of the canvas and then the other, thereby rolling the sample back and forth. After thoroughly mixing in this manner, the sample will be formed in a conical pile and reduced in quantity by quartering.

Crushing of increments.

41. Whenever the different increments of samples are collected throughout some considerable period of time, each increment may be crushed as soon as taken and the pieces of coal and impurities broken sufficiently small to permit two or three reductions of the total accumulated sample before further crushing is necessary.

Special moisture sample.

42. In the reduction of the gross sample to the sample for transmittal to the laboratory the gross sample will unavoidably lose moisture. To determine the moisture content in the coal delivered, a separate special moisture sample must be taken. This will be accumulated by placing in a hermetically sealed receptacle small parts of the freshly taken increments of the gross sample.

Moisture samples discretionary.

43. The collection of special moisture samples shall be discretionary with the Government. Special moisture samples will be taken if in the opinion of the Government inspector the coal contains moisture in excess of the amount guaranteed by the contractor. The special moisture samples will be taken so as to represent the coal with respect to the moisture contained at time of weighing.

Extended deliveries.

44. If deliveries extend over any considerable period, what would otherwise be a gross sample may be worked down in successive stages to samples of a size suitable for transmittal to the laboratory, and the samples representing the several equal parts of a delivery may be analyzed and the several analyses averaged, or the several samples may later be mixed (at the delivery point or in the laboratory and reduced to one sample) and one analysis made.

X. ANALYSIS.

45. Immediately on receipt of a sample in the laboratory it will be analyzed and tested by the Government in accordance with the method recommended by the American Chemical Society, and by the use of a bomb calorimeter.

XI. CAUSES FOR REJECTION.

(A) BITUMINOUS.

Character.

46. All coal delivered during a fiscal year is expected to be of the same character as that specified by the contractor. It should, therefore, be supplied as nearly as possible from the same mine or group of mines.

Quality.

47. It is important that the standards furnished with bids shall not establish a higher value than can be actually maintained under the terms of the contract. In this connection it should be recognized that the small "mine samples" usually indicate a coal of higher economic value than that actually delivered in carload lots, because of the care taken to separate extraneous matter from the coal in the "mine samples." It is evident, therefore, that it will be to the best

interests of the contractor to furnish a correct description with average values of the coal offered, as a failure to maintain the standard he establishes will result in deductions from the contract price, and may cause a cancellation of the contract, whereas deliveries of coal of higher grade than quoted will be paid for at an increased price per ton.

Unsatisfactory fuel.

48. Coal containing percentages of volatile matter or sulphur higher than the limits indicated under "Description of Coals Desired," or having a moisture content in excess of that guaranteed, or containing percentages of ash greater than indicated in the column "Maximum limits for ash" in the table in the section entitled "Price and Payment," or failing to give satisfactory results because of excessive clinkering or a prohibitive amount of smoke, or proving for any other cause to be an undesirable fuel, will be subject to rejection, and the Government will have the right to cause the contractor to remove such coal at no cost to the Government. Such event may result in the Government purchasing in the open market, or through competitive bidding, such quantity of coal to supply the deficiency caused by such failure, or annulling the contract by giving notice in writing to that effect to the contractor, and the Government, in its discretion, purchasing such coal in the open market or by contract upon competitive proposals; the contractor to remain liable for all damages sustained by the United States on account of such failure, including the difference, if any, between the cost of purchasing and delivering said coal and the price at which the contractor agreed to furnish it. The contract price must be understood to be the corrected price per ton based upon analysis, as hereinafter described under the section entitled "Price and Payment." If it shall be impracticable to cause the contractor to remove coal subject to rejection, the Government may use such rejectable coal, deducting penalties as determined under the section entitled "Price and Payment," and may in addition, as circumstances warrant in the opinion of the Government, deduct a further penalty of twenty-five (25) per cent of the amount of the bill, based on the tonnage of the coal under question and at the contract price per ton.

(B) ANTHRACITE PEA AND BUCKWHEAT NO. 1.

Character and quality.

49. It is understood that the coal delivered during the year will be of the same character as that supplied for the service test made to determine the final award of the contract. It is important, therefore, that the quality and character of the coal furnished for service tests shall not establish a standard higher than can be maintained, for a failure to maintain the standard may cause the Government to purchase coal in the open market, and charge against the contractor being made for the excess in cost over the bid price; or the contract may be cancelled.

Unsatisfactory fuel.

50. If during the life of the contract, coal is delivered that gives unsatisfactory service results as compared with the results obtained on the coal tested to determine the final award of the contract, or if the analysis and tests on a "dry-coal" basis of a sample taken from coal used during any consecutive twenty-four (24) hours show an ash content of 5 per cent more or a yield of British thermal units of 5 per cent less, than was shown by the sample taken to determine the final award of the contract, or should any of the other results of analysis or test indicate coal of inferior quality or of different character than that designated in the contract, then the lot of coal from which the coal fired for twenty-four (24) hours was obtained will be subject to rejection, and the Government will have the right to cause the contractor to remove at no cost to the Government the remaining coal of the delivery. Such

event may result in the Government purchasing in the open market, or through competitive bidding, such quantity of coal to supply the deficiency caused by such failure, or annulling the contract by giving notice in writing to that effect to the contractor, and the Government, in its discretion, purchasing such coal in the open market or by contract upon competitive proposals; the contractor to remain liable for all damages sustained by the United States on account of such failure, including the difference, if any, between the cost of purchasing and delivering said coal and the price at which the contractor agreed to furnish it. If it shall be impracticable to cause the contractor to remove coal subject to rejection, the Government may use such rejectable coal, and may deduct a penalty of 25 per cent of the amount of the bill based on the tonnage of the delivery under question and at the contract price per ton, if in the opinion of the Government the circumstances warrant the deduction.

Unacceptable coal.

51. Coal containing an undue percentage of fine coal, bone, slate, dirt, or moisture will not be accepted by the Government inspector.

XII. PRICE AND PAYMENT.

(A) BITUMINOUS COAL.

Prompt payment.

52. Payment will be made promptly upon receipt of a report from the Bureau of Mines on the quality of the coal under consideration. The Bureau of Mines will furnish such report in not more than fifteen (15) days after the receipt of the sample or samples.

Determination of price.

53. Payment for coal specified in the proposal will be made upon the basis of the price therein named, corrected as follows for variations in heating value, ash, and moisture from the standards specified in the contract (see section 48 for an additional deduction on coal subject to rejection):

(*a*) Considering the coal on a "dry-coal" basis, no corrections in price will be made for variations of 2 per cent or less in the number of British thermal units from the guaranteed standard. When the variation in heat units exceeds 2 per cent of the guaranteed standard, the correction in price will be a proportional one and will be determined by the following formula:

$$\frac{\text{B. t. u. delivered coal ("dry-coal" basis)}}{\text{B. t. u. ("dry-coal" basis) specified in contract}} \times \text{bid price} = \text{price resulting from B. t. u. correction.}$$

For example, if coal delivered on a contract guaranteeing 14,000 British thermal units on a "dry-coal" basis at a bid price of $3 per ton shows by calorific test results varying between 13,720 and 14,280 British thermal units, there will be no price correction. If, however, the delivered coal shows by calorific test 14,360 British thermal units on a "dry-coal" basis for example, the price for this variation from the contract guarantee is, by substitution in the formula:

$$\frac{14,300}{14,000} \times \$3 = \$3.064$$

The correction will be figured to the nearest tenth of a cent.

(*b*) For all coal that by analysis contains less ash on a "dry-coal" basis than the percentage specified herein, a premium of 2 cents per ton for each whole per cent less will be paid. An increase in the ash content of 2 per cent above the standard established by the contractor will be tolerated without exacting a penalty. When such excess is greater than 2 per cent, deductions will be made in accordance with the following table:

Ash as established in proposal.	No deduction for limits below—	Cents per ton to be deducted.							Maximum limits for ash.[a]
		2	4	7	12	18	25	35	
		Percentages of ash in dry coal.							
Per cent.	*Per cent.*								
4	6, inclusive	6.01- 7.00	7.01- 8.00	8.01- 9.00	9.01-10.00	10.01-11.00	11.01-12.00	12.01-13.00	11
5	7, inclusive	7.01- 8.00	8.01- 9.00	9.01-10.00	10.01-11.00	11.01-12.00	12.01-13.00	13.01-14.00	12
6	8, inclusive	8.01- 9.00	9.01-10.00	10.01-11.00	11.01-12.00	12.01-13.00	13.01-14.00	14.01-15.00	13
7	9, inclusive	9.01-10.00	10.01-11.00	11.01-12.00	12.01-13.00	13.01-14.00	14.01-15.00	15.01-16.00	14
8	10, inclusive	10.01-11.00	11.01-12.00	12.01-13.00	13.01-14.00	14.01-15.00	15.01-16.00	16.01-17.00	14
9	11, inclusive	11.01-12.00	12.01-13.00	13.01-14.00	14.01-15.00	15.01-16.00	16.01-17.00	17.01-18.00	15
10	12, inclusive	12.01-13.00	13.01-14.00	14.01-15.00	15.01-16.00	16.01-17.00	17.01-18.00		16
11	13, inclusive	13.01-14.00	14.01-15.00	15.01-16.00	16.01-17.00	17.01-18.00	18.01-19.00		16
12	14, inclusive	14.01-15.00	15.01-16.00	16.01-17.00	17.01-18.00	18.01-19.00	19.01-20.00		17
13	15, inclusive	15.01-16.00	16.01-17.00	17.01-18.00	18.01-19.00	19.01-20.00			18
14	16, inclusive	16.01-17.00	17.01-18.00	18.01-19.00	19.01-20.00	20.01-21.00			19
15	17, inclusive	17.01-18.00	18.01-19.00	19.01-20.00	20.01-21.00				19
16	18, inclusive	18.01-19.00	19.01-20.00	20.01-21.00					20
17	19, inclusive	19.01-20.00	20.01-21.00	21.01-22.00					21
18	20, inclusive	20.01-21.00	21.01-22.00	22.01-23.00					22

[a] These limits are used in determining rejectable coal, see section 48, marginal heading "Unsatisfactory fuel."

As an example of the method of determining the deduction in cents per ton for coal containing ash exceeding the standard by more than 2 per cent, suppose coal delivered on a contract guaranteeing 10 per cent ash on the "dry coal" basis shows by analysis between 14.01 and 15 per cent (both inclusive), or, for instance, 14.55 per cent, the deduction according to the table is 7 cents per ton (reading to the right on line beginning with 10 per cent on the extreme left, which in this case is the standard, to the column containing "14.01–15," the deduction at the top of this column is seen to be 7 cents).

NOTE.—If the ash standard is an uneven percentage, the table will be revised in order to determine deductions on account of excessive ash. For example, if the ash standard is 6.53 per cent, each percentage value beginning with 6 in the left-hand column and all figures in the line reading to the right of 6 will be increased by 0.53. There would be no deduction then in price for ash in delivered coal up to and including 8.53 per cent, while for coal having an ash content, for instance, between 11.54 and 12.53 per cent the deduction would be 12 cents per ton.

(*c*) The price will be further corrected for moisture content in excess of the amount guaranteed by the contractor, the deduction being determined by multiplying the price bid by the percentage of moisture in excess of the amount guaranteed. The correction will be figured to the nearest tenth of a cent. For example, if coal delivered on a contract guaranteeing 3 per cent moisture with bid price of $3.50 per ton shows by analysis 4.65 per cent moisture, the bid price is multiplied by 1.65 (the excess moisture), which gives 5.8 cents ($0.058) as the deduction per ton.

[$3.50×(4.65−3.00=1.65 per cent)=5.775 cents.]

(B) ANTHRACITE PEA AND BUCKWHEAT NO. 1.

Prompt payment.

54. Payment, which will be promptly made upon completion of an order, will be based upon the contract price, provided the coal is not subject to rejection (see section entitled "Causes for rejection").

XIII. INFORMATION TO BE SUPPLIED.

55. Estimated quantity of coal required, ——— tons (2,000 or 2,240 pounds).

56. (The point of delivery, method of delivery, capacity and facilities for storage, etc., are here furnished.)

57. The bidder must insert in the blank spaces below the information called for on the coal he proposes to furnish, without which information the proposal will be informal:

(A) BITUMINOUS.[a]

(*a*) Commercial name of the coal .....

(*b*) Name of the mine or mines .....

(*c*) Location of the mine or mines (town, county, State) .....

.....

(*d*) Railroad on which mine is located .....

[a] Bidders are cautioned against specifying higher standards than can be maintained, for to do so may result in the bid being rejected (sec. 8), or may result in rejection of delivered coal or cancellation of the contract and the Government purchasing coal in the open market and charging against the contractor the difference in cost (see secs. 47 and 48).

(*e*) Name or other designation of the coal bed or beds .....
.....
(*f*) Name of operator of mine or mines .....
(*g*) British thermal units per pound of "dry coal" .....
(*h*) Percentage of ash in "dry coal" .....
(*i*) Percentage of sulphur in "dry coal" .....
(*j*) Percentage of volatile matter in "dry coal" .....
(*k*) Moisture in coal "as received" .....
(*l*) Additional description of coal as deemed of importance by the bidder.....
.....
(*m*) Price per ton of ..... pounds (this price is understood to be the bid price per ton, see section 53 for method of determining price for delivered coal) .....

(B) ANTHRACITE, PEA, AND BUCKWHEAT NO. 1.

(*a*) Coal field .....
(*b*) District .....
(*c*) Colliery and location.....
(*d*) Name and address of operator .....
(*e*) Railroad upon which colliery is located .....
(*f*) Name or other designation of the coal bed or beds .....
.....
(*g*) Additional description of coal as deemed of importance by the bidder.....
.....
(*h*) Price per ton of ..... pounds:
Pea .....
Buckwheat No. 1 .....

The SECRETARY OF THE ..... DEPARTMENT:

The undersigned hereby propose .. to furnish and deliver to .....
..... the coal as above specified, and at the price stated above, subject to all the conditions of the specifications and proposal, for the fiscal year commencing on the 1st of July, ... and ending on the 30th of June, ...

The undersigned ha.. read the specifications and proposal and agree.. to comply therewith in every particular.

Signature of each member of the firm and firm name. If a corporation, its name, and signature of the officer authorized to sign for the corporation, together with a copy, under seal of his authority to sign; also, the name of the State in which incorporated. .....
.....
.....
.....

Doing business under the firm name of.....

Place of business, ...
.....

NOTE.—Owing to the difficulty in deciphering signatures, a typewritten copy of same should be attached.

SPECIFICATIONS AND PROPOSALS FOR ANTHRACITE FURNACE (BROKEN), EGG, STOVE, AND CHESTNUT COAL.

I. PROPOSALS.

1. (Same as section 1, page 52.)
2. (Same as section 2, page 52.)

II. ADDRESS OF PROPOSALS.

3. (Same as section 3, page 52.)

III. PROPOSALS—GUARANTY.

Signature. 4. (Same as section 4, page 52.)

Cash or certified check. 5. (Same as section 5, page 52.)

IV. CONTRACTOR'S BOND.

Sureties. 6. (Same as section 6, page 52.)

May be waived. 7. (Same as section 7, page 53.)

V. RESERVATIONS.

Rejection and annulment. 8. (Same as section 8, page 53.)

Estimated quantity. 9. (Same as section 9, page 53.)

Lowest bids may not be considered. 10. (Same as section 11, page 53.)

Failure to contract. 11. (Same as section 12, page 53.)

Contracts nontransferable. 12. (Same as section 13, page 54.)

Default. 13. (Same as section 14, page 54.)

VI. DESCRIPTION OF COAL DESIRED.

Character. 14. The coal must be the best quality Pennsylvania white-ash anthracite.

Quality. 15. It must be well screened, practically free from dirt, must not contain undue percentages of moisture, slate, or bone, and must equal in quality and preparation the best white-ash anthracite produced. The coal must not contain more than the following percentages of ash on the "dry-coal" basis:

Size of coal.	Ash in "dry coal."
Furnace	10 per cent.
Egg	11 per cent.
Stove	12 per cent.
Chestnut	14 per cent.

Information to be supplied by bidder. 16. (Same as section 20, page 54.)

VII. AWARD.

Consideration. 17. Bids will be considered on each item separately, and in determining the award of the contract consideration will be given to the

results obtained with coals furnished on previous Government contracts by the respective bidders, as well as to the price per ton specified.

Lowest bids may not be considered.

18. If it should appear to the best interests of the Government to do so, the right will be reserved to award the contract for supplying coal at a price higher than that named in a lower bid or in lower bids.

VIII. DELIVERY.

Quantity.

19. (Same as section 27, page 56).

Rapidity.

20. (Same as section 28, page 56).

Notices.

21. (Same as section 29, page 56).

Hours.

22. (Same as section 30, page 56).

Weighing.

23. (Same as section 31, page 56).

IX. SAMPLING.

Method.

24. Samples which may be taken (see section 27) will be collected in accordance with the methods and principles which are a part of the specifications generally used by the Government for the purchase of coal under specifications and as set forth in U. S. Bureau of Mines Bulletin 63.

X. ANALYSIS.

25. Samples will be analyzed and tested by the Government in accordance with the method recommended by the American Chemical Society.

XI. CAUSES FOR REJECTION.

Unsatisfactory fuel.

26. If the percentage of bone, dirt, slate, or smaller sizes in a given coal exceeds the percentage required by the best preparation, or if the given coal fails to give satisfactory results because of excessive clinkering, or excessive ash-pit refuse, it will be subject to rejection, and the Government will have the right to cause the contractor to remove such coal at no cost to the Government. The Government may then purchase coal in the open market and make charge against the contractor for the excess in cost of coal so purchased. The Government inspector will not accept coal if in his opinion it is rejectable, or if it contains an undue percentage of moisture, and he will have the right to refuse to accept coal from a source other than that specified by the bidder under section 32, unless the contractor may have received permission from the Government to substitute said coal.

Ash deductions.

27. If it is impracticable for the Government to cause the contractor to remove coal that produces excessive ash-pit refuse, samples may be taken during the consumption of such coal, and if on analysis the ash content on the "dry-coal" basis is equal to or greater than the following percentages for the respective sizes of coal, ten (10) per cent of the bid price of the coal delivered and unused at the time of sampling will be deducted:

Size of coal.	Ash in "dry coal," per cent.
Furnace	14
Egg	15
Stove	16
Chestnut	18

Open-market purchases.

28. The delivery of unsatisfactory fuel may result in the Government purchasing in the open market, or through competitive bidding,

such quantity of coal to supply the deficiency caused by such failure, or annulling the contract by giving notice in writing to that effect to the contractor, and the Government, in its discretion, purchasing such coal in the open market or by contract upon competitive proposals; the contractor to remain liable for all damages sustained by the United States on account of such failure, including the difference, if any, between the cost of purchasing and delivering said coal and the price at which the contractor agreed to furnish it.

XII. PRICE AND PAYMENT.

29. Payment will be promptly made on completion of an order and will be based on the contract price, provided the coal is a satisfactory fuel. If the coal is unsatisfactory (see sections 26 and 27), and it is impracticable for the Government to cause the contractor to remove such coal, payment will be promptly made on ninety (90) per cent of the total amount of the bill, based on the contract price and the tonnage of the order, the ten (10) per cent being withheld to cover the deduction provided for under section 27, "Ash deductions."

XIII. INFORMATION TO BE SUPPLIED.

30. (Same as section 55, page 62.)

31. (Same as section 56, page 62.)

32. The bidder must insert in the blank spaces the information called for on the coal he proposes to furnish, without which information the proposal will be informal:

(*a*) Coal field.....

(*b*) District.....

(*c*) Colliery and location.....

(*d*) Name and address of operator.....

(*e*) Railroad upon which colliery is located.....

(*f*) Name or other designation of the coal bed or beds.....

(*g*) Price per ton of ..... pounds:

Furnace.....

Egg.....

Stove.....

Chestnut.....

The SECRETARY OF THE ..... DEPARTMENT:

The undersigned hereby propose.. to furnish and deliver to .... the coal as above specified and at the price stated above, subject to all the conditions of the specifications and proposal, for the fiscal year commencing on the 1st of July, ..., and ending on the 30th of June, ...

The undersigned ha.. read the specifications and proposal and agree.. to comply therewith in every particular.

Signature of each member of the firm and firm name. If a corporation, its name and signature of the officer authorized to sign for the corporation, together with a copy, under seal, of his authority to sign; also the name of the State in which incorporated. .....

Doing business under the firm name of .....

Place of business .....

.....

NOTE.—Owing to the difficulty in deciphering signatures, a typewritten copy of same should be attached.

Bulletin 64

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

THE TITANIFEROUS IRON ORES

IN THE UNITED STATES

THEIR COMPOSITION AND ECONOMIC VALUE

BY

JOSEPH T. SINGEWALD, JR.

WASHINGTON
GOVERNMENT PRINTING OFFICE
1913

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PREFACE.

One of the problems that confronts American ironmasters is the efficient reduction of magnetite containing high percentages of titanium. From time to time the discovery of enormous bodies of workable titaniferous ore has been reported, and the available literature dealing with the geologic occurrence and the economic value of the titaniferous magnetites in the United States contains a mass of information that in part is based on incomplete observation and even on hearsay.

Although the supplies of nontitaniferous iron ore that are available in the United States are still to be measured by millions of tons, yet the approaching exhaustion of many high-grade deposits, the general acceptance by blast furnaces of ores lower in iron than those utilized 10 years ago, and the probability of even lower-grade ores being used in increasing quantities indicate that some large deposits of ore would be contributing to the needs of the country were it not for their titanium content. In an attempt to ascertain just what the economic possibilities of the larger deposits of titaniferous iron ore may be, the Bureau of Mines took up the study of the possible utilization of these ores through some concentrating process. The conduct of this investigation was assigned to Mr. J. T. Singewald, jr., and the results are presented in this report. Mr. Singewald was directed to pay especial attention to the possibility of utilizing titaniferous ores at a profit with present methods and devices and under existing conditions. He was authorized to study in detail the physical structure and chemical composition of the ores, as bearing on the problem of utilization, but not to undertake any detailed investigation of the origin and geologic distribution of the deposits.

It is to be regretted that the results of Mr. Singewald's careful study of the ores are chiefly negative. Some deposits of titaniferous iron ore are clearly not as extensive nor as high in iron as they were reported to be. Moreover, examination of many samples by metallographic methods has demonstrated that although a large part of the titanium in these ores is in the form of ilmenite, much of this ilmenite is far more intimately associated with the magnetite than has generally been supposed. Part of the magnetite and ilmenite occurs in such large and distinct aggregates that their separation by a magnetic concentration after fine crushing is practicable, but in by far

the larger number of samples examined and in practically all the ore bodies that are known to be large enough and rich enough in iron to be of much importance, most of the ilmenite occurs as such fine intergrowths in the mass of the magnetite that a complete separation of the two minerals by any process based on difference of physical properties still seems impractical. Crushing even to 200 mesh would not insure a clean separation of the two minerals. Consequently the problem of utilizing titaniferous magnetites involves the application of chemical rather than physical methods. In short, the problem is not one of eliminating the titanium by milling but of reducing the ores directly by some smelting process.

Iron ores containing over 1 per cent of titanium are troublesome to smelt in the blast furnace. Though Rossi and other investigators have demonstrated the possibility of eliminating titanium through the formation of relatively fluid slags consisting of complex titanosilicates, there seems to be no inclination on the part of the iron-masters to adopt furnace practice to these conditions. However, as Mr. Singewald points out, it is practicable to use in blast-furnace mixtures small proportions of the concentrates from a magnetic separation of the titaniferous ores with little change in present methods and without danger of choking a furnace. The use of electricity in smelting offers other possibilities. The reduction of titaniferous ores in electrically heated furnaces and the production of iron-titanium alloys by a direct process are problems of such importance that the Bureau of Mines, in connection with its investigations of the treatment of ores and of more efficient methods of smelting, is undertaking experiments with these ores. The results of these experiments will be published in future reports of the bureau.

J. A. HOLMES.

THE TITANIFEROUS IRON ORES IN THE UNITED STATES; THEIR COMPOSITION AND ECONOMIC VALUE.

By JOSEPH T. SINGEWALD, Jr.

INTRODUCTION.

The term "titaniferous magnetite" is used to designate those magnetic ores of iron that carry more than 2 or 3 per cent of titanium. Large and easily workable deposits of these ores occur in different parts of the world, and have attracted attention for many years. Under present furnace practice, however, the smelting of these ores is both difficult and expensive, and for that reason they are not accepted by furnaces, so that the deposits are practically worthless. The problem of finding a feasible method of utilizing these ores has naturally been attractive, and numerous attempts have been made to solve it.

Experiments to that end embrace two lines of investigation. The one that has been the more frequently tried is the elimination of the titanium from the ores, so as to make possible their smelting in the usual way. The other is the devising of a method of smelting by which titaniferous ores may be reduced as economically as are nontitaniferous ores. The former method involves a physical process, the latter a chemical process.

The measure of success attainable by the first method depends entirely on the relations between the iron and the titanium in the ore. Views in regard to these relations have been divergent. Some investigators have believed the titanium to occur as a part of the magnetite molecule, in which case, of course, any process of mechanical separation would be impossible. Others have believed that the titanium occurs as the mineral ilmenite, and that the ores are aggregates of magnetite and ilmenite. In this case a separation of the two minerals is theoretically possible. Magnetite is highly magnetic; ilmenite is nonmagnetic or only feebly magnetic. Crushing and magnetic concentration should separate them. Numerous experiments in magnetic separation have been made with various degrees of success, but rarely has more than a very imperfect separation been obtainable. In other words, some of the titanium exists in a separable form and some in an inseparable form, and the ratio between the two forms is

different in different ores. The whole question as to the nature of the ores remains little understood.

The use of the metallographic microscope seemed to offer a means of determining the relations that exist between the titanium and the iron in these ores. In the winter of 1911, in the capacity of Henry E. Johnston scholar at the Johns Hopkins University, the author took up the study of the problem along these lines. The necessary apparatus and laboratory facilities were provided by the geological department of the university. Material for a preliminary study was very kindly furnished by Prof. J. F. Kemp from the collection of titaniferous ores made by him and contained in the collections at Columbia University. This material was supplemented by ores obtained from Mr. W. L. Cumings, of the Bethlehem Steel Co.

A study of this material yielded results warranting a more thorough investigation of the problem, and this was made possible through the Bureau of Mines, with which the author became connected in June, 1911. The summer of 1911 was spent in visiting the principal localities in the United States at which titaniferous iron ores occur, for the purpose of making systematic collections of material. The following winter was spent in a study of the collected material in the geological laboratory of the Johns Hopkins University where the author continued as a Johnston scholar. This report is an account of the results obtained.

The author wishes to express his appreciation to those who made possible the carrying out of this investigation and to those who rendered assistance during its progress.

THE UTILIZATION OF TITANIFEROUS IRON ORES.

As stated above, the problem of the utilization of the titaniferous iron ores can be attacked along two lines—first, by the use of metallurgical processes, involving the direct reduction of the ores; second, by the use of milling methods, involving the elimination of titanium from the ores. The progress that has been made along these lines is briefly stated in the following pages.

THE SMELTING OF TITANIFEROUS IRON ORES.

SMELTING IN THE BLAST FURNACE.

Notwithstanding extensive discussions of the subject from time to time, the possibilities of smelting titaniferous iron ores in the blast furnace are still undetermined. That such ores have been smelted and excellent pig iron made from them is well known. It is equally well established that in many such cases the consumption of fuel has been so great as to make the cost of the process excessive. At present there is a prejudice, justifiable or not, against the use of ores carrying

more than 1 per cent of titanium. This prejudice is based chiefly on the belief that the ores require an undue amount of fuel to reduce them, and that they cause the accumulation of infusible titanium compounds that ultimately choke the furnace.

A historical review of the use of titaniferous iron ores shows that their use has gradually declined. In the early days of the Catalan forge and small charcoal furnaces it did not seem to make any particular difference whether an ore was titaniferous or not. Many deposits of titaniferous ores were worked and smelted without difficulty. This fact no doubt is explainable in part by the condition of the iron industry at that time. Some of the arguments that are used against the ores at present would have been of little weight in those days. The enterprises were small and the production of iron was of far greater moment than the cost of production. The early furnaces and forges were surrounded by virgin forests, and the amount of charcoal required was not considered an important factor. Later, as the furnaces increased in size and the expense of procuring charcoal became greater, the quantity of fuel required to produce a ton of iron had to be taken into consideration. As the higher the titanium content of an ore, the lower the possible iron content, the use of titaniferous ores was of itself equivalent to the use of lower-grade ores. Further, the slags produced from titaniferous ores in ordinary furnace practice are less fluid, and hence the furnace had to be run at higher temperatures. This method of operation caused the reduction of the titanium, a result that was favorable to the formation of infusible accretions of titanium in the furnace. Thus the use of titaniferous ores tended naturally to a high fuel consumption, and at times to the choking of the furnace. The prejudice against the ores became so great that finally no furnace would knowingly use them.

However, because the deposits of titaniferous ore are so numerous and so large, and the ores so low in phosphorus and sulphur, they have never ceased to attract attention and numerous experiments have been conducted looking to a means of utilizing them. During the sixties and seventies of the last century the subject was repeatedly discussed in the Journal of the British Iron and Steel Institute. Attention was called to the fact that these ores had been worked and were then being worked in a number of furnaces in Norway, Sweden, and England. At some of those furnaces the entire charge consisted of titaniferous ores, at others they were mixed with nontitaniferous ores. With the evolution of the modern blast furnace, however, all of these attempts have gradually been discontinued. A century ago there were 15 small charcoal furnaces in operation at Taberg in Sweden, but for a number of years the deposit has been wholly abandoned.

Probably the most extensively used of these ores in the United States were those of Lake Sanford, N. Y. A furnace with a daily capacity of 3 or 4 tons was built there in 1840. The furnace was subsequently enlarged, and a second furnace with a capacity of 12 to 15 tons was built. This remained in operation until 1856. The natural difficulties due to the inaccessible location of the enterprise were of themselves almost insuperable, so that its duration is rather remarkable apart from any difficulties due to the highly titaniferous character of the ores.

Two blast furnaces were erected on Bay St. Paul in 1873 by the Canadian Titanic Iron Co. to utilize the titaniferous ores of that part of the St. Lawrence region,[a] but were operated for only a few years. Under the most favorable circumstances 3,040 to 3,792 pounds of charcoal per ton of iron was required, and when the ore and the limestone had become wet or covered with ice, as much as 6,400 pounds was required. Harrington states that the experience in Sweden has been that in many cases more than twice as much charcoal is necessary than is required to smelt ordinary magnetites.

In most cases it seems that no attempt was made to adapt the furnace charges to the nature of the ores. It will be interesting, therefore, to review some of the results that have been obtained when such attempts were made.

David Forbes, who was connected with charcoal ironworks in southern Norway that used titaniferous ores, states [b] that the experience of the Scandinavian ironmasters has shown that the only objection to their use is that they require a large amount of charcoal, and hence their use is not profitable where other ores free from titanium can be obtained at a reasonable rate. For this reason he found it unprofitable to smelt alone an ore carrying 42.04 per cent Fe and 15.10 per cent TiO_2, but could readily use such an ore with equal parts of a nontitaniferous ore. In the attempts to smelt it more easily, he says, his predecessor, supposing that a volatile compound of silicon and titanium would be formed, fluxed the ore with increasing charges of stamped quartz until a cast iron was obtained so highly charged with silicon that "it flowed from the furnace like porridge." Forbes went to the other extreme in fluxing with lime so as to form a titanate of lime and obtained unsatisfactory results. Subsequently he found that by using both quartz and limestone as fluxes silicotitanates of much more fusible character were formed and more satisfactory results were obtained. When the amount of titanium in the ore did not exceed 8 per cent, or was reduced to 8 per cent by admixing other

[a] Harrington, B. J., Notes on the iron ores of Canada and their development: Geol. Survey of Canada, Rept. for 1873–74, pp. 249–251.

[b] Forbes, David, On the composition and metallurgy of some Norwegian titaniferous iron ores: Chem. News, 1868, pp. 275–276.

ores, no difficulty was experienced in working the ore cleanly and profitably.

An attempt made by the Norwegian Titanic Iron Co. to smelt highly titaniferous ores at Norton, England, met with failure after a few years' trial. W. M. Bowron, who was connected with the company as chemist, states [a] that "the process, regarded as a process, was a perfect success; but the enormous quantity of fuel required, the small quantity of iron in the ore, and the cost and uncertainty of importation militated seriously against its commercial success." The ores used by the company were composed of 36.33 per cent Fe and 39.20 per cent TiO_2. Nearly 3 tons of charcoal were required to produce 1 ton of iron. The smelting of such an ore is obviously not a fair test of what can be done with average titaniferous iron ores. A nontitaniferous magnetite carrying such a low percentage of iron would not be used in a blast furnace without previous concentration. The results are important in that they show the possibility of smelting even the highest titanium-bearing ores successfully, though not economically, in the blast furnace.

A very thorough study of the problem has been made in this country by Rossi. In 1890 he published [b] a review of what had been accomplished in the way of smelting titaniferous iron ores both in this country and abroad. Rossi's review, however, adds no essential facts to what has already been said. Three years later he published [c] the results of some experiments that he conducted. His experiments, made along the lines adopted by Forbes, were designed to discover easily fusible titanate slags of low viscosity. He first experimented with the formation of compounds that could be easily fused and liquefied. He then mixed ore charges to produce slags of such composition to determine whether pig iron could be made to separate from them. The results of these experiments are given in detail in the paper cited; it will be sufficient to state here that he obtained the best results with a tribasic slag, in which the bases were lime, magnesia, and alumina. The experiments were first conducted in a small furnace without forced draft; subsequently a small blast furnace was erected. The latter yielded pig iron and slags that showed good fluidity and fusibility, and there was no choking of the furnace with infusible titanium compounds. The ores experimented on ranged from 18.70 to 20.03 per cent TiO_2.

In 1895 the experiments were repeated by Rossi [d] on a larger scale in a furnace with a daily capacity of about 3 tons, no difficulties being experienced in running the furnace. These experiments showed that

[a] Bowron, W. M., The practical metallurgy of titaniferous ores: Trans. Am. Inst. Min. Eng., vol. 11, 1883, pp. 159–164.

[b] Rossi, A. J., Titanium in blast furnaces: Jour. Am. Chem. Soc., vol. 12, 1890, pp. 90–117.

[c] Rossi, A. J., Titaniferous ores in the blast furnace: Trans. Am. Inst. Min. Eng., vol. 21, 1893, pp. 832–864.

[d] Rossi, A. J., The smelting of titaniferous ores: Iron Age, 1896, vol. 1, pp. 354–356, 464–469.

the most refractory slags could be melted without reducing silica, and hence much less titanic acid. Under the conditions of the experiments he says that furnaces can not be troubled with titanium deposits. "The deposits consist of cyano-nitride of titanium, which supposes for its formation not only the reduction in the furnace of titanic acid to titanium but the highest temperature and other conditions." He further says: "It may be argued that circumstances may so occur in the running of a blast furnace smelting any kind of nontitaniferous ores that would lead to an obstruction whose removal would require forcing the heat and the pressure of the blast, and that these circumstances in the special case of titaniferous ores would be favorable to the formation of titanium deposits by the reduction of the titanic acid." His reply to this is that the action of the blast furnace is so well understood and in such competent hands at the present time that such conditions are very rare; but if they should arise, it would be easy to change the charge to nontitaniferous ores until the obstruction has been removed and the furnace brought back to normal condition.

In order to become familiar with the behavior of his furnace under normal charges, it was first run for a time on Lake Superior ores; these were then gradually replaced by titaniferous ores and the flux changed from calcite to a dolomite carrying 12 to 14 per cent of magnesia. The titaniferous ores consisted of four-fifths to five-sixths Sanford or Millpond ore and one-fifth to one-sixth Cheney Pond ore. Rossi's account of the results is given below:

In order to judge the relative economy of these three runs under as nearly as possible similar conditions, we will compare the amounts of fuel and stone required per unit of pig metal when the furnace gave the greatest production in each case. This supposes, indeed, for the kind of ores or mixtures of ores considered, the most favorable conditions of running for each. By making precisely the same ample allowance of time in each case for stone and coke before each maximum cast of 24 hours as chargeable to that cast, we found the figures given below:

Results of Rossi's blast-furnace tests of different ores, based on unit quantity of pig iron when greatest production was obtained.

RUN NO. 1 WITH NONTITANIFEROUS HEMATITES FROM LAKE SUPERIOR.

Stone, unit quantity required	1.15
Coke, unit quantity required	2.15
Fe in ores, per cent	62
Maximum cast of iron in 24 hours, pounds	4,600

RUN NO. 2 WITH MIXTURE OF HEMATITE AND TITANIFEROUS ORES.

Stone, unit quantity required	1.19
Coke, unit quantity required	2.20
Fe in ores, per cent	56
Maximum cast of iron in 24 hours, pounds	5,035

RUN NO. 3 WITH TITANIFEROUS ORES FROM THE ADIRONDACKS.

Stone, unit quantity required	0.95
Coke, unit quantity required	1.99
Fe in ores, per cent	52
TiO_2 in ores, per cent	20
Maximum cast of iron in 24 hours, pounds	6,735

Hence, to say the least, the titaniferous ores, under the same conditions of furnace running, did not require any more fuel per unit of pig metal than excellent non-titaniferous ores. Really they required decidedly less, and the production of the furnace was increased considerably.

Summing up the results of his experiments, Rossi concludes:

1. As anybody who may desire to make the experiment can verify, titanic acid can form definite compounds, perfectly fusible, if properly fluxed, containing as much as 35 to 40 to 50 per cent of titanic acid, with alumina, lime, and magnesia as bases, and admissible as slags in blast-furnace work. Larger percentages still, such as 65 per cent, can enter into a compound and it remains fusible. The objections to the smelting of titaniferous ores on account of the refractory character of the slags are not sustained by our practice or that of others, or by direct experiments on the properties of these compounds.

2. In running a furnace under special conditions of temperature and pressure of blast no troubles have been experienced from the titanium deposits. We never observed any in our blast-furnace tests, and none are mentioned by Dr. Forbes in his practice in England and Norway.

3. If these special conditions of lower heat, considered more favorable in smelting these ores, are held to imply against them a waste of fuel, it is a question whether this is not offset by the smaller amount of cinder to melt, the lesser quantity of fluxes necessary and their indirect effect upon the productive capacity of the furnace, as well as the greater value of the pig metal obtained for specific and numerous applications. This is without taking into account the possibility of not submitting to it by a rapid driving and forcing the production, conditions which, to judge from our tests, could be easily realized with these ores.

However conclusive Rossi's experiments may appear, no attempts have been made to carry them out on a commercial scale; and the titaniferous ores remain in hopeless disrepute so far as their use in the ordinary blast furnace is concerned.

SMELTING IN THE ELECTRIC FURNACE.

Electric smelting of iron ores is still in its infancy, and its full possibilities have not yet been demonstrated. Where conditions are favorable it has met with considerable success. Electric smelting is of special interest in this connection since there is a confident feeling in many quarters that herein lies the hope of the utilization of the titaniferous ores. However, our knowledge in this regard has not yet reached the point where any very definite conclusions can be drawn.

Experiments in the electrical reduction of titaniferous iron ores have been conducted in Canada. About seven years ago, Haanel [a] carried out a series of experiments for the Canadian Government at Sault Ste. Marie in the smelting of Canadian iron ores by the electrothermic process, titaniferous iron ores being used in one of the experiments. Unfortunately this run, which was the last one made, was not as complete as the others. The furnace lining was in very bad condition, but as the time for making the experiments had nearly expired it could not be repaired and the run had to be stopped before figures as to output had been obtained. The ore used contained 17.82 per cent TiO_2, and the furnace charge consisted of 400 pounds of ore, 100 pounds of charcoal, 50 pounds of limestone, and 50 pounds of fluorspar. The slag produced was very fluid, and the statement is made that "likely the fluorspar in the charge could have been reduced considerably or omitted altogether."[b]

Abstracts from an article on this subject, recently published by Stansfield,[c] who has experimented along these lines, are quoted below:

Although the possibility of the electric smelting of iron ores for pig iron can not be definitely settled in a general way, there appears to be every probability that a process which yields steel instead of pig iron would be commercially possible in favorable localities. Steel could be made by smelting titaniferous iron ores in an electric furnace with a somewhat smaller amount of charcoal than would be needed for making pig iron, and the resulting metal could be transferred in the molten condition to an electrical steel-refining furnace, where it could be made into finished steel. The cost of producing the crude steel from the ore would be little more than that of producing pig iron, and the cost of refining the steel would be only a moderate addition to this, while the price obtainable for the finished product would be decidedly higher than could be obtained for pig iron.

The writer has been associated with Mr. J. W. Evans in the development of a process in which the reduction of titaniferous ores to metal and the subsequent refining of the steel is carried out in a single furnace, and while the details of the furnace construction and certain parts of the operation have not yet been fully worked out the results have been so encouraging that it appears to be worth while to develop the process on a small working scale. With regard to the results so far obtained, it may be stated that titaniferous ores obtained from the Orton mine and containing about 7 per cent of titanium have been smelted directly to steel, and that such steel, containing about 1 per cent of carbon, has been found to possess unusually good qualities as a tool steel. The cost of smelting these ores and refining the steel can be determined fairly accurately from the figures published in regard to the electrical smelting of iron ores to pig iron and the electrical refining of molten steel. These figures are more dependable than data obtained from the small-scale experiments that have so far been made. The writer intends, however, in the near future to

[a] Haanel, Eugene, Report on the experiments made at Sault Ste. Marie, Ontario, under Government auspices in the smelting of Canadian iron ores by the electrothermic process: Canada Dept. of Mines, Mines Branch, 1907, pp. 85–86.

[b] Haanel, Eugene, Preliminary report on the experiments made at Sault Ste. Marie, Ontario, under Government auspices in the smelting of Canadian iron ores by the electrothermic process: Canada Dept. of Mines, Mines Branch, 1906, p. 19.

[c] Stansfield, Alfred, Electric smelting of titaniferous ores: Can. Min. Jour., vol. 33, 1912, pp. 448–449.

make further experiments on the reduction and refining of the steel in order to obtain direct information as to the cost of the process.

There does not appear to be any definite limit as regards the percentage of titanium in ores suitable for this process, but the cost of smelting will, of course, increase with the proportion of titanium, both on account of the cost of fluxing it and on account of the smaller proportion of steel obtained. It seems probable, therefore, that the process will be applied to ores that do not contain more than 5 to 10 per cent of titanium. Ores somewhat richer than this in titanium may be concentrated magnetically, so as to obtain a product which shall be at once richer in iron and poorer in titanium, and it may even be possible in some cases that the tailings from such a concentration process may be sufficiently rich in titanium to be smelted for ferrotitanium. Some ores containing too much titanium for steel making are not amenable to magnetic concentration, and these may be smelted with suitable admixture of nontitaniferous ores. It has been found that the electric furnace is more suitable than the blast furnace for the treatment of pulverized ores and that magnetic concentrates can be smelted alone or with the addition of a moderate proportion of coarser ore.

The use of titaniferous ores for making tool steel would appear to be very advantageous commercially, but can not be expected to provide for a large consumption of these ores, and it is of interest to consider whether the process can be applied to the production of steel on a larger scale. One purpose for which this process would appear to be very suitable is for the production of steel castings. A good deal of difficulty is experienced in the production of steel castings which can be relied upon as being entirely sound, and ferrotitanium is used in some cases for producing soundness in steel castings. It seems very probable, therefore, that steel made directly from titaniferous ores will be particularly suitable for the production of steel castings, and this will afford a larger field for the electric smelting of these ores. With regard to the use of this process on a still larger scale—for the production of steel rails, for example—the writer would say that there appears to be no definite reason why rails should not be made by this process, but would not care to speak definitely with regard to this until the process has been in operation on a working scale.

At least there has been opened a new field of investigation, in which the solution of the problem of the utilization of the titaniferous iron ores may be found.

ELIMINATION OF TITANIUM BY MAGNETIC CONCENTRATION.

The fact that many titaniferous magnetites are granular aggregates of magnetite and ilmenite is quite apparent to the naked eye in many of the coarser-grained ores. The highly lustrous rougher surface of the ilmenite can be easily distinguished from the duller black cleavage faces of the magnetite. This difference is brought out more plainly if the specimen is immersed for a time in concentrated hydrochloric acid, which acts on the magnetite but does not affect the ilmenite. The magnetite becomes still duller and pitted as a result of the solvent action of the acid, whereas the ilmenite remains unchanged. The relations between the magnetite and the ilmenite grains are brought out still more clearly if the specimen is ground down to a plane surface, polished, and treated with hydrochloric acid.

The polished ilmenite grains retain their luster, whereas the polished magnetite loses its luster and assumes a dull black. Specimens of ore that have been treated in this way are shown in Plate II, *B, C, D, E,* and *F* (p. 50), Plate VIII, *B* (p. 106), and Plate IX, *B* (p. 110).

Newland[a] applied this treatment to some of the Adirondack ores. Commenting on the results obtained, he says:

> In the specimens that have been examined the magnetite and ilmenite are distinguishable without difficulty. There is a clear separation of the particles and no notable tendency toward intergrowth or inclusion on the part of either. The boundaries are sharp. Both minerals belong to the same order of crystallization, though the ilmenite seems to have begun to form somewhat earlier than the magnetite. The latter being commonly in excess constitutes the groundmass through which the ilmenite is more or less regularly distributed in grains of fairly even size.

A great many of the bright ilmenite grains have been isolated and tested with a magnet, and in every case were found to be practically nonmagnetic. Dull grains of magnetite similarly tested have in every case been found to be highly magnetic. Further the specimens mentioned above bear out Newland's statement that there is "no notable tendency toward intergrowth or inclusion on the part of either." Although there are numerous small grains of ilmenite scattered over the surface of the magnetite, they form an insignificant part of the total visible ilmenite; and nearly all of the ilmenite grains in the coarse granular ores are of sufficient size to be separated from the magnetite by crushing to a size allowable in actual commercial operations. If all of the titanium in titaniferous magnetites were contained in these ilmenite grains, magnetic concentration ought to give concentrates practically free from titanium. Some results that have actually been obtained in attempting to make a magnetic separation of the titanium are presented below:

RESULTS OF MAGNETIC-SEPARATION EXPERIMENTS.

The results of magnetic separations of foreign titaniferous magnetites are tabulated first; these are followed by a tabulation of the results of separations of ores from the United States. The latter results are also discussed subsequently. Pope[b] made three tests of ores from eastern Ontario. The experiments were made with a Wetherell magnetic concentrator at Newark, N. J. The ore was crushed so that it would pass through a 40-mesh screen but not through a 60-mesh screen.

No separation of Eagle Lake ore was possible. As soon as the current was strong enough to pick up a single particle all were picked up. This ore analyzed 64.68 per cent Fe and 6.41 per cent TiO_2.

[a] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, 1908, pp. 151–152.

[b] Pope, F. J., Investigation of magnetic iron ores from eastern Ontario: Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 372–405.

The results with Pine Lake ore are tabulated below:

Results of magnetic concentration of Pine Lake ore.

	Current.	Weight.	Fe.	TiO_2.
		Grams.	*Per cent.*	*Per cent.*
Ore	Single dry cell	380	43.38	13.52
Heads		272	56.45	18.10

The result of the concentration was to increase the metallic iron by ten thirty-thirds and the titanic oxide by eleven thirty-thirds.

The results with Chaffey ore were as follows:

Results of magnetic concentration of Chaffey ore.

	Current.	Weight.	Fe.	TiO_2.
	Amperes.	*Grams.*	*Per cent.*	*Per cent.*
Ore	0.25	720	53.0	7.4
Heads		640	56.1	5.6
Tails		73	27.0	29.9

Here the iron was increased 6 per cent and the titanium reduced 32 per cent by the concentration.

The results of many magnetic concentrations, principally of Norwegian ores, have been tabulated by Vogt,[a] as shown below:

Results of magnetic concentration of titaniferous iron ores from Heindalen.

Test designation.	Composition of ore.	Yield of concentrate.	Composition of concentrate.
	Per cent.	*Per cent.*	*Per cent.*
b	7.08 TiO_2 44.66 Fe 0.95 S 0.013 P	49.02	1.49 TiO_2 68.51 Fe. 0.097 S. 0.006 P.
b c	6.20 TiO_2 35.85 Fe 0.023 P	35.46	1.30 TiO_2 67.73 Fe. 0.004 P.
c c	4.90 TiO_2 27.65 Fe 0.042 P	23.98	0.72 TiO_2 69.90 Fe. 0.003 P.

[a] Vogt, J. H. L., Norges Jernmalmforekomster: Norges Geologiske Undersokelse 51, 1910, pp. 31–34. A German review by Vogt is contained in the Zeitschr. prakt. Geol., 1910, pp. 62–64.

[b] Represents a large-scale test made by a company in Stockholm.

[c] Represents laboratory test made by Prof. Henry Louis.

Results of magnetic-concentration tests by Farup.

Test designation.	Composition of ore.	Effects of crushing ore to 1 mm.		Effects of crushing ore to 0.1 mm.	
		Yield of concentrate.	Composition of concentrate.	Yield of concentrate.	Composition of concentrate.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
A	7.15 TiO_2 35.07 Fe	46.51	.....	34.97	0.40 TiO_2 69.60 Fe.
B	8.48 TiO_2 42.61 Fe	* 65.79	.....	50.00	0.96 TiO_2 68.40 Fe.
C1	4.32 TiO_2 21.33 Fe	28.01	.....	20.92	1.03 TiO_2 66.34 Fe.
C2	6.60 TiO_2	54.05	3.97 TiO_2	33.22	1.22 TiO_2.
C3	6.30 TiO_2	58.82	2.65 TiO_2	42.02	1.16 TiO_2 67.59 Fe.
C4	7.21 TiO_2	50.50	4.23 TiO_2	36.63	1.47 TiO_2 67.4 Fe.
C5	6.81 TiO_2	64.10	3.33 TiO_2	47.62	1.53 TiO_2 65.68 Fe.
D1	9.03 TiO_2 49.3 Fe 0.81 S 0.017 P	75.19	5.00 TiO_2 62.24 Fe	63.29	1.24 TiO_2 66.52 Fe.
D2	8.21 TiO_2 46.44 Fe 1.02 S 0.014 P	60.60	.....	50.25	1.20 TiO_2 68.70 Fe. 0.45 S. 0.008 P.
D3	7.80 TiO_2 51.20 Fe	69.93	4.09 TiO_2 61.38 Fe	63.69	2.35 TiO_2 64.59 Fe.
E	5.48 TiO_2 62.0 Fe	90.90	2.39 TiO_2 68.17 Fe	87.72	1.60 TiO_2 69.65 Fe.
F	32.9 TiO_2 45 Fe e	24.39	4.27 TiO_2 63.65 Fe	19.23	1.75 TiO_2 68.1 Fe.
G	15.04 TiO_2 50 Fe e	59.17	.....	47.17	2.37 TiO_2 68.60 Fe.
H	13.55 TiO_2 40 Fe e	65.36	10.01 TiO_2 47.34 Fe	47.85	4.81 TiO_2 60.99 Fe.
I1	13.49 TiO_2 50 Fe e	81.97	.....	53.48	6.26 TiO_2 59.07 Fe.
I2	10.87 TiO_2 50 Fe e	71.94	.....	41.67	6.53 TiO_2 55.16 Fe.
K	14.10 TiO_2	77.52	14.52 TiO_2	67.57	9.18 TiO_2.
L	7.78 TiO_2 49.52 Fe	76.92	8.02 TiO_2 53.72 Fe	65.79	7.9 TiO_2 57.2 Fe.
M	9.80 TiO_2 41.2 Fe	71.43	12.98 TiO_2	62.89	10.80 TiO_2.

* Approximate.

Tests A, B, C, etc., represent ores from different deposits; C1, C2, C3, etc., represent ores from different places in the same deposit. A, B, and C represent ores from Heindalen, D from Rödsand, F from Laxedal in the Ekersund-Soggendal district, G from Solnor, and I from Lied. The sources of the others are not given.

Commenting on the results of these tests, Vogt says that "the so-called titanomagnetite in these tests consists of a mechanical mixture of magnetite and ilmenite. Whether the magnetite contains a small amount of TiO_2 in chemical combination is an open question. It is not probable, however, as the investigations show that the finer the crushing, the greater the quantity of titanic acid removed." In the cases of tests G, H, and I he regards the intergrowth to have been somewhat finer than in tests A, B, C, and D; and in K, L, and M he considers that much of the intergrowth was of microscopic size. Whether at the time of the solidification of the magmas primary

magnetite and primary ilmenite crystallized, or whether at that time a magnetite formed that at high temperature contained ilmenite in solid solution, the ilmenite subsequently segregating at lower temperatures, he says can not yet be decided.

In addition to the above tests, Vogt gives the results of several tests made on magnetic sands. The results of one of these are presented below:

Results of magnetic-concentration tests of magnetic sand from Lofoten Islands, Norway.

[Size of sand grains—73.3 per cent less than one-quarter mm. but greater than one-sixth mm.; 26.7 per cent less than one-sixth mm.]

Determined constituents of sand.	Amount.	Condition of sand.					
		Not crushed.		Crushed to less than ⅛ mm.		Powdered.	
		Concentrate.	Tailings.	Concentrate.	Tailings.	Concentrate.	Tailings.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	
TiO_2	13.20	5.03	26.90	3.14	26.7	2.5	.....
Fe	56.55	67.70	43.56	69.24	43.59	70.33	.....
P	.06	.045	.....	.026	.....	.....	.....
S	.007	.002	.....	.002	.....	.....	.....
SiO_2	3.52	.85	.....	.250	.....	.....	.....
Percentage of whole	.....	53.8	46.2	50.5	49.5	47.7	52.3

The tests were made by a company in Stockholm, Sweden, as were also the first two in the table following. The third was made under the direction of Prof. Vogt with a Gröndal separator No. 5. Concentrate 1 was made without crushing the sand. This concentrate was then crushed to one-sixth mm. and a second separation made from it.

Results of magnetic-concentration tests of sands from lower St. Lawrence, Canada.

Quality of sand.	Composition of sand.	Amount of concentrate 1 obtained.	Composition of concentrate 1.	Amount of concentrate 2 obtained from 1.	Composition of concentrate 2.	Composition of tailings 2.
		Per cent.		*Per cent.*		
Lean	5.7 TiO_2, 18.6 Fe	10.9	4.4 TiO_2, 65.6 Fe	89.4	2.9 TiO_2, 69.8 Fe	30.1 Fe.
Rich	18.6 TiO_2, 54.8 Fe	42.2	2.3 TiO_2, 69.6 Fe	96.6	1.5 TiO_2, 70.5 Fe	44 Fe.
———	7.9 TiO_2, 29.4 Fe	26.5	3.9 TiO_2, 66.3 Fe	95	2.0 TiO_2, 69.6 Fe	37 Fe.

The results of magnetic concentrations of titaniferous iron ores of the United States are presented below. They are a tabulated summary of the results discussed in a subsequent section of this report. The first table gives the results of tests previously made and the second of tests made by the author in the course of this investigation.

In the author's tests small samples of ore were crushed in a diamond mortar to pass through a 50-mesh screen having an opening of 0.3 mm. The crushed material was sorted into two sizes with a 100-mesh screen having an opening of 0.15 mm. Each size was then subjected to magnetic separation under water with a hand magnet. The resulting concentrates and tailings were analyzed for Fe and Ti by A. C. Fieldner, of the Bureau of Mines. The composition of the original ore was calculated from the ratio of concentrates to tailings.

Results of magnetic separations previously made.

Source of ore.	Size of screen passed through.	Determined constituents of ore.		Determined constituents of concentrate.		Determined constituents of tailings.	
		TiO_2.	Fe.	TiO_2.	Fe.	TiO_2.	Fe.
		Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Sanford, N. Y.				4.00	62.66	47.50	38.86
Do.	40-mesh	14.0	60.58	8.93	60.43	45.23	45.78
Do.				9.66	60.60	32.22	42.84
Do.[a]	16-mesh			5.25	63.00		
Do.	40-mesh			[b] 5.90	[b] 65.02		
Do.				11.00	61.04		
Tuscarora, N. C.		15.35	41.95	1.27	67.60	16.20	21.63

[a] From concentrate of ore mentioned just above.

[b] Average of two analyses.

Results of magnetic separations made by the author.

Source of ore.	Determined constituents of ore.		Effect of crushing ore to 0.3 mm.					Effect of crushing ore to 0.15 mm.		
			Yield of concentrate.	Determined constituents of concentrate.		Determined constituents of tailings.		Yield of concentrate.	Determined constituents of concentrate.	
	TiO_2.	Fe.		TiO_2.	Fe.	TiO_2.	Fe.		TiO_2.	Fe.
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
Lincoln Pond, N. Y.	5.88	64.66	96.0	4.82	66.08	31.45	30.59	95.6	5.12	66.27
Sanford, N. Y.	19.02	55.07	78.2	11.15	61.23	47.20	32.99	75.9	10.47	61.23
Millpond, N. Y.	18.02	54.47	83.2	13.18	58.89	43.03	32.74	85.3	12.28	59.17
Tuscarora, N. C.	12.82	58.07	71.5	4.25	67.76	34.32	33.76	70.1	3.64	68.41
Iron Lake, Minn.	24.30	53.30	93.3	23.15	54.93	40.37	30.59	92.0	23.58	54.13
Iron Mountain, Wyo.	21.75	52.31	84.2	18.30	56.75	40.12	28.74	83.6	17.25	56.75
Do.	19.47	54.48	77.7	12.58	60.76	43.47	32.61	76.0	10.32	61.97
Do.	21.85	51.93	87.3	20.03	55.35	34.33	28.43	86.5	30.03	55.44
Shanton Ranch, Wyo.	21.08	51.50	84.7	17.85	55.05	38.93	31.83	82.2	17.85	55.63
Do.	19.98	52.98	84.1	17.22	56.93	34.58	31.96	78.6	16.72	57.30
Iron Mountain, Colo.	14.50	51.96	86.0	13.50	56.93	20.97	21.42	50.4	13.10	57.77

The table following is compiled from the two preceding tables. It gives the amount of titanium per unit of iron in the ore and in the concentrates, and where known the size to which the ore had been crushed. From these ratios the percentage reduction of titanium per unit of iron is calculated. It is recognized that the results of such a calculation are not strictly comparable. The percentage of iron given by analysis is the total iron content. Some of this iron is present in the iron-bearing minerals of the gangue, and is not asso-

ciated with the titanium, and these minerals are practically removed from the concentrates. The ratio of titanium to iron in the concentrates is therefore the ratio of the titanium to the iron with which it is actually associated, whereas the ratio of titanium to iron in the ore is the ratio of the titanium to the iron with which it is associated plus the iron contained in the gangue minerals. The iron contained in the gangue is removed in the magnetic concentration of ordinary magnetic ores, and this process does not bear on the problem of eliminating titanium from titaniferous magnetite. The presence of iron in the gangue lowers the calculated ratio of titanium to iron in the ore, and hence the calculated percentage of titanium removed per unit of iron is less than is actually the case. The leaner the ore the greater the discrepancy. For this reason, the percentage of iron present in the crude ore is included in the table.

Summary of results of magnetic separations.

FOREIGN ORES.

Source of ore.	Ore.		Concentrate.		
	Fe.	Ti per unit Fe.	Maximum size of grain.	Ti per unit Fe.	Reduction of Ti per unit Fe.
	Per cent.	*Per cent.*	*Mm.*	*Per cent.*	*Per cent.*
Eagle Lake	64.68	0.0595	[a] 0.3	0.0595	0
Pine Lake	43.38	.1870	[a] .3	.1924	[b] 3
Chaffey	53.0	.0838	[a] .3	.0537	36
Heindalen *a*	44.66	.0951	.....	.0130	86
Heindalen *b*	35.85	.1038	.....	.0115	89
Heindalen *c*	27.65	.1063	.....	.0062	94
Heindalen A	35.07	.1223	.1	.0035	97
Heindalen B	42.61	.1194	.1	.0084	93
Heindalen C1	21.33	.1215	.1	.0093	92
Rödsand D1	49.3	.1099	.1	.0103	90
Rödsand D2	46.44	.1061	.1	.0105	90
Rödsand D3	51.20	.0914	.1	.0218	76
E	62.0	.0530	.1	.0138	74
Laxedal F	[a] 45	.4387	.1	.0154	96
G	[a] 50	.1805	.1	.0207	89
H1	[a] 40	.2032	.1	.0473	77
I1	[a] 50	.1619	.1	.0636	61
I2	[a] 50	.1304	.1	.0710	46
L	49.52	.0943	.1	.0829	12
Lofoten	56.55	.1401	.2	.0272	81
St. Lawrence (lean)	18.6	.2161	.2	.0249	89
St. Lawrence (rich)	54.8	.2036	.2	.0123	94
St. Lawrence	29.4	.1612	.2	.0172	89

UNITED STATES ORES.

Sanford	60.58	0.1388	[a] 0.3	0.0886	36
Tuscarora	41.95	.2195	.....	.0113	95
Lincoln Pond	64.66	.0546	.15	.0464	15
Sanford	55.07	.2072	.15	.1026	50
Millpond	54.47	.2005	.15	.1245	38
Tuscarora	58.07	.1324	.15	.0319	76
Iron Lake	53.30	.2735	.15	.2614	5
Iron Mountain, Wyo	52.31	.2495	.15	.1824	27
Do	54.48	.2144	.15	.0999	53
Do	51.93	.2544	.15	.2168	15
Shanton	51.50	.2456	.15	.1925	22
Do	52.98	.2263	.15	.1750	23
Iron Mountain, Colo	51.96	.1674	.15	.1361	18

[a] Approximate. [b] Increase.

The results of magnetic concentrations just cited show that the relations between the iron and the titanium in the titaniferous magnetites are not as simple as a megascopic examination of the ores would indicate. It is evident that a large part of the titanium is contained in the magnetite itself. The variation in the degree of separation obtained in the different cases shows further that the percentage of ilmenite in separable form and that present in inseparable form is not constant, but varies over a wide range. The only advantage to be derived from magnetic concentration of these ores, in so far as the titanium is concerned, is that in some cases the percentage of titanium can be brought low enough to make practicable the mixing of the concentrates with two, three, or four parts of nontitaniferous ores, whereby the titanium content would be made negligible.

MICROSTRUCTURE OF TITANIFEROUS MAGNETITE.

PREVIOUS INVESTIGATIONS.

It has been shown that in most titaniferous magnetites that have been subjected to magnetic concentration a large percentage of the titanium occurs in the magnetite in a form not separable from the magnetite by that process and not visible to the naked eye. That seemingly homogeneous magnetite crystals can contain a high percentage of titanium has been recognized for many years, and the manner in which it occurs in the magnetite has been the subject of considerable speculation. Fifty years ago Knop [a] analyzed some magnetite octahedra from Meiches in the Vogelsberg, and found 24.95 per cent of TiO_2; and Kemp [b] says that O. A. Derby has mentioned to him the occurrence of natural loadstone in Brazil with 20 per cent of TiO_2. A sample of seemingly pure magnetite picked from a specimen of Lake Sanford ore contained 11 per cent of TiO_2.[c]

In 1858 Rammelsberg [d] expressed the view that the titanium-bearing magnetites consist of mixtures of titanium-free octahedral magnetite and rhombohedral ilmenite. Since that time a number of workers have recognized microscopic intergrowths of magnetite in ilmenite.

Though the intergrowths of ilmenite and magnetite are far more common, intergrowths of rutile and magnetite were discovered about the same time. Thus in 1877 Seligmann [e] described regular intergrowths of rutile in magnetite crystals from the Alp Lercheltiny in

[a] Knop, A., Ueber titansäure—haltigen Magneteisenstein: Liebig's Ann. chem. pharm., vol. 123, 1862, p. 352.

[b] Kemp, J. F., The titaniferous iron ores of the Adirondacks: U. S. Geol. Survey 19th Ann. Rept., pt. 3, 1899, p. 386.

[c] See p. 76.

[d] Rammelsberg, C., Ueber die Zusammussetzung des Titaneisens, sowie der rhomboëdrisch und oktaëdrisch Krystallisirten Eisenoxyde überhaupt: Pogg. Ann., vol. 104, 1858, pp. 497–552.

[e] Seligmann, G., Mineralogische Notizen: Groth Zeitschr. für Kryst. und Min., vol. 1, 1877, pp. 340–342.

Binnenthal, Switzerland. About the time that the microscopic intergrowths of magnetite in ilmenite were first discovered Cathrein[a] described a regular microscopic intergrowth of rutile in magnetite analogous to the macroscopic intergrowth described by Seligmann. He found that on dissolving the magnetite in hydrochloric acid there was left behind an aggregate of minute rutile microlites arranged parallel to the three edges of an octahedral face and intersecting each other at an angle of 60°. The magnetite itself appeared black and opaque, wholly homogeneous, and even great magnification failed to reveal the intergrown rutile.

Neef,[b] describing a diabase, says:

> Accessory constituents are pyrite and a black ore showing on the surface of the section distinct systems of lines cutting each other at acute angles which seem to indicate intergrowths of ilmenite lamellæ. On etching the section with hot hydrochloric acid the ore readily dissolved, leaving behind an undecomposed latticework skeleton of ilmenite. Is this perhaps an intergrowth of specularite or magnetite with ilmenite lamellæ cutting each other at acute angles?

The same observation was made by Küch[c] a few years later in studying some West African rocks. He says in the description of a plagioclase-olivine-augite rock:

> On treating a thin section with hydrochloric acid the magnetite goes into solution. There remains behind, however, a latticework skeleton formed of laths which must be ilmenite, and which as such are lacking in the unetched section. This relation seems to point to intergrowths of ilmenite and magnetite, the latter being dissolved through the treatment with acid, so that only the ilmenite lamellæ remain. The same phenomenon has already been observed by Neef in a Swedish diabase.

About the same time a similar conclusion was reached by Teall[d] from a somewhat different consideration. His article was read in June, 1884, but refers to the one by Küch just quoted. He found an opaque iron oxide which carried considerable TiO_2, and was yet attracted by a weak bar magnet. The analysis and the physical properties of the substance, he believed, agreed with the assumption of a mixture of magnetite and ilmenite. To corroborate such an assumption he refers to the work previously cited, and to the fact that in the whin sill the substance alters into leucoxene along two sets of parallel planes. He concludes, therefore, that an intergrowth of magnetite and ilmenite actually exists.

The first mention of a microscopic intergrowth of magnetite and ilmenite was made by Becke[e] in 1886. He found that spheres of

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[a] Cathrein, A., Ueber die mikroskopische Verwachsung von Magneteisen mit Titanit und Rutil: Groth Zeitschr. für Kryst. und Min., vol. 8, 1884, pp. 321–329.

[b] Neef, M., Ueber seltenere krystallinische Diluvialgeschiebe der Mark: Zeitschr. Deutsch. geol. Gesell., vol. 34, 1882, p. 479.

[c] Küch, R., Beitrag zur Petrographie des westafricanischen schiefergebirges: Tschermak's Min. Pet. Mitt., vol. 6, 1885, p. 129.

[d] Teall, J. J. H., On the chemical and microscopic characters of the whin sill: Quar. Jour. Geol. Soc. of London, vol. 40, 1884, p. 657.

[e] Becke, F., Aetzversuche an Mineralen der Magnetitgruppe: Tschermak's Min. Pet. Mitt., vol. 7, 1886, pp. 222–233

magnetite produced from octahedra from Pfitsch, Tyrol, Austria, were not homogeneous, but filled with tablets of ilmenite which were intercalated in the magnetite parallel to the octahedral faces. A year later Cathrein[a] described the same phenomenon more fully in the case of magnetite crystals from the Zillerthal, Tyrol, Austria. He found scattered over the octahedral faces of the magnetite small laths, 1 to 2 mm. long and 0.5 mm. wide, which on closer examination proved to be the edges of black metallic tablets. Chemical investigation showed that the magnetite itself was wholly free from titanium and that the tabular inclusions were ilmenite. A study of the orientation of the tabular crystals of ilmenite within the magnetite showed that the basal plane of the ilmenite lay parallel to the octahedral face of the magnetite; that is, the position assumed was that in which similar axes of symmetry lay parallel.

In 1892, Rosenbusch[b] stated that Latermann communicated to him the finding of ilmenite intergrowths in the magnetite of the nepheline basalt of the Katzenbuckel, near Heidelberg, Germany. If the magnetite in the thin section were dissolved in hydrochloric acid, there remained a network of ilmenite lamellæ which in sections parallel to the faces of (111) cut each other at an angle of 60° and in sections parallel to the faces of (001) at 90°.

Adams[c] observed two distinct iron ores in the ore particles of the anorthosite of the Morin district, in Canada, on examining thin sections in reflected light. One specimen he describes as having a streaked appearance, due to the fact that one mineral cut the other in a simple or double group of broken laths. On treating the microscope slide with hydrochloric acid the one mineral dissolved, but the laths remained, showing an intergrowth of magnetite with ilmenite, or at least, he says, with a titaniferous iron ore.

In an augite mica diorite from the island of Cabo Frio, Rosenbusch[d] found particles of titaniferous magnetite, which on treatment with hydrochloric acid left behind a network of brownish specular ilmenite.

Lacroix[e] ascribes the titanium content of titaniferous magnetite both to intergrowths of ilmenite and magnetite and to the presence of titanomagnetite. In intergrowths of ilmenite and magnetite the tabular crystals of ilmenite are distributed in the crystals of magnetite parallel to the octahedral face, so that the two minerals have a common ternary axis. The manner of intergrowth is the

[a] Cathrein, A., Verwachsung von Ilmenit mit Magnetit: Groth Zeitschr. für Kryst. und Min., vol. 12, 1887, pp. 40–46.

[b] Rosenbusch, H., Mikroskopische Physiographie der petrographischen wichtigen Mineralien, 1892. p. 287.

[c] Adams, F. D., Ueber das Norian oder Ober-Laurentian von Canada: Neues Jahrb. Min., Geol., Paleon., 1893, B. B. 8, p. 445.

[d] Rosenbusch, H., Mikroskopische Physiographie der massigen Gesteine, 1896, p. 246.

[e] Lacroix, A., Mineralogie de la France et de ces Colonies, vol. 3, 1901–1909, pp. 287–288.

same as that in the case of hematite and magnetite. He says: "It is very probable that many titaniferous magnetites owe their titanium content to similar groupings of a physical nature and are consequently different from titanomagnetite." Titanomagnetite[a] he calls magnetite in which a small part of the ferric oxide has been replaced by Ti_2O_3.

The intergrowths of ilmenite and rutile in magnetite are briefly described by Mügge,[b] who, however, adds nothing to what has been stated above. The chief interest of Mügge's paper is that it discusses at some length the physical and chemical laws governing regular intergrowths of different minerals.

All of the information in regard to the intergrowths of titanium minerals in the magnetite of titaniferous magnetites that had been obtained up to this time came in the course of mineralogical and petrographic studies. No effort had been made to apply the method of attack pointed out by these results to a study of the titaniferous ores themselves. The first investigation of such a character was carried out by Hussak[c] on some Brazilian titaniferous magnetites. Hussak investigated titaniferous magnetite derived both from stringers and veinlets in acidic rocks and from magmatic segregations in basic rocks. Polished plane surfaces of the ores were prepared and etched with hydrochloric acid. The ilmenite, being unaffected by the acid, retained its luster, whereas the magnetite was attacked by the acid, producing a dull-black surface against which the bright ilmenite was easily recognized. He found that the ores consisted in general of aggregates of magnetite and ilmenite, and that the magnetite contained minuter intergrowths of ilmenite, in places regularly intergrown according to the manner already described, and in places irregularly intergrown. The regularly intergrown lamellæ were also found to vary considerably in size and abundance. Five microphotographs of these intergrowths accompanied his article, which so far as known to the author are the only ones that have been published up to the present time.

The lamellar intergrowths of ilmenite were also observed by Warren[d] in the Rhode Island ores, and described as follows:

> On etching the polished surface with hydrochloric acid, a delicate reticulate structure is developed in the ore. With a direct illumination under the microscope, this structure is seen to consist of an intersecting series of bright-gray, narrow bands or lamellæ inclosing dull-black depressions. This structure extends everywhere through the ore except over occasional small areas which have a smooth surface and are iden-

[a] Lacroix, A., op. cit., vol. 4, 1910, p. 317.

[b] Mügge, O., Die regelmässigen Verwachsungen von Mineralen verschiedener Art: Neues Jahrb. Min., Geol., Paleon., 1903, B. B. 16, pp. 335–475.

[c] Hussak, E., Über die Mikrostructur einiger brasilianischer Titanmagneteisensteine: Neues Jahrb. Min., Geol., Paleon. vol. 1, 1904, pp. 94–113.

[d] Contributions to the geology of Rhode Island. I, Notes on the history and geology of Iron Mine Hill Cumberland by B. L. Johnson. II, The petrography and mineralogy of Iron Mine Hill, Cumberland, by C. H. Warren: Am. Jour. Sci., vol. 25, 1908, pp. 15–16.

tical with the bands in appearance and presumably in composition. The lamellæ are frequently observed to cross at angles of nearly or exactly 60 degrees; again they form a nearly rectangular grating. This structure, taken in connection with the well-known tendency of ilmenite to develop a reticulate structure parallel to the rhombohedral planes, suggests at once that the lamellæ are ilmenite inclosing the more easily dissolved magnetite. The change in direction of the bands in different parts of the ore matrix is the only indication of the existence of individual grains in the ore.

The ratios derived from the rock analysis show conclusively that the two molecules, $R^{III}TiO_3$ and $R^{II}R^{III}_2O_4$, are present, and in about equal proportions. The magnetic susceptibility of particles of the ore, free from olivine and feldspar, was found to be intermediate between that of pure magnetite and ilmenite. It appears, therefore, that the ore matrix consists essentially of an intimate intergrowth of magnetite and ilmenite with an occasional grain of what is probably pure ilmenite. The parallel intergrowth of members of the spinel family with ilmenite has long been familiar to mineralogists, and a similar intergrowth has been suggested by a number of petrographers to explain the composition of the "titaniferous magnetites" of many rocks. Quite recently Hussak has shown by the study of etched surfaces that intergrowths of ilmenite and spinel minerals, particularly magnetite, are very general in the titaniferous ores of Brazil. He also gives a full bibliography of the literature on the subject. It would be extremely interesting to extend the study of titaniferous magnetites still further and ascertain to what extent, if at all, an intergrowth like the one described in the present instance exists in other occurrences. There is still, in spite of the considerable amount of investigation which has been carried on upon "titaniferous magnetites," much uncertainty regarding the exact relations of these two mineral molecules when occurring together.

In order to bring out the relations between the larger ilmenite grains and the magnetite grains of the titaniferous ores of the Adirondacks, Newland[a] prepared polished sections which were etched with acid, and he gives three photographs taken with an ordinary camera. It seems, however, that he did not examine the surfaces under magnification and hence overlooked the ilmenite intergrowths within the dull magnetite areas.

METALLOGRAPHIC METHOD OF INVESTIGATION.

The microstructure of the titaniferous magnetites has been studied in this investigation by the application of what is known as the metallographic method. It is a method that has long been employed in metallurgy for investigating the microstructure of the metals and alloys. Polished surfaces of the substance to be examined are prepared and treated with etching fluids, and the structure brought out in this way is observed with a microscope specially constructed for using reflected light, and known as a metallographic microscope.

The instrument used in this work was the Leitz metallographic microscope in the geological laboratory of the Johns Hopkins University. This instrument is so constructed that the objective is inverted and the illumination, which is provided by a small arc light, is reflected upward from below. The object can therefore be

H., Geology of the Adirondacks magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 3,

placed face downward on the stage, and the polished surface is at once at right angles to the axis of the microscope. This arrangement has the advantage that it does away with the necessity of mounting the specimens, a detail of considerable importance when a large number of specimens are to be studied.

To prepare a polished section, a small piece of ore was chipped or sawed off the hand specimen. The most convenient size was found to be about a half inch square. Elongate pieces were avoided, as it was difficult to keep a plane surface on them in grinding. A little pressure on one side produced a curved surface, and the smallest curvature made the polishing more tedious and necessitated a constant readjustment of the focus of the microscope. The chip was ground down to a plane surface on an ordinary lap wheel, coarse powders being first used, followed by the use of the finest carborundum or emery, until a plane surface had been produced. For the final stages of preparation, a small lap wheel was constructed, run directly by a motor by means of a friction drive. The speed of the wheel could be regulated by changing the gear of the friction drive and by means of a starting box. Duplicate wheels were provided, so that only one kind of powder was used on each wheel. To obtain a smoother surface than was obtained on the large wheel with the finest carborundum, the carborundum was levigated and the finest sediment used on the small wheel. Still better results were obtained when instead of proceeding directly to the polishing stage, the surface was rubbed for a time on a soft whetstone. For this purpose the black stone used in polishing marble was found satisfactory. The final polishing was done with putty powder on a wheel covered with ribless broadcloth.

The polished surface thus prepared was treated with strong hydrochloric acid. The length of treatment necessary to bring out the structure depended greatly on the character of the ore. Coarsely crystallized ore is more quickly acted upon by the acid than fine-grained aggregates. If the etching was not continued long enough, the structure was not well brought out and much detail was lost; on the other hand, if it proceeded too far, the difference in relief became too great and all the details could not be brought into focus at one time. The most satisfactory way was occasionally to remove the specimens from the acid and observe them under the microscope; as soon as the desired contrast was obtained, the etching was stopped.

In preparing the microphotographs accompanying the report a peculiar difficulty was encountered. All the ilmenite intergrowths and the contacts between the larger ilmenite grains and the magnetite showed a blurred effect on one side. Thus, in the case of the linear network, one side of the lines would be perfectly sharp and the other gradually fade away. The author is greatly indebted to Prof. A. H.

Pfund, of the department of physics at the Johns Hopkins University, for the solution of this difficulty. These lines, as well as the other intergrowths, instead of showing on the ground glass as single bright lines showed in the form of spectra; and it was these spectra that were being photographed. Prof. Pfund suggested boring holes in the carbons and filling them with finely pulverized salt and then introducing between the arc and the microscope a filter filled with a solution of potassium bichromate of sufficient strength to cut out all bands of the spectrum beyond the yellow sodium band. In this way what was practically a bright-yellow monochromatic light was produced, as the faint red in this spectrum has no action on the photographic plate. With the aid of this light the blurred effect was eliminated and sharp photographs were produced.

RESULTS OF METALLOGRAPHIC STUDY.

If a polished section of a titaniferous magnetite is treated with hydrochloric acid, the etched surface reveals to the naked eye certain more apparent features of the ore. It is at once seen that the greater part of the section is made up of three principal types of surfaces. Rather light-colored nonmetallic surfaces are at once recognized as the gangue minerals. Dark gangue minerals, which in the unaltered ore can scarcely be distinguished from the ore minerals, magnetite and ilmenite, are bleached and easily recognized. Other parts of the section have remained unaltered and retain their bright metallic surfaces. These consist of grains of ilmenite. Still other areas, made up of grains of magnetite, assume a somewhat dull-black appearance. A closer scrutiny of the magnetite and the ilmenite areas often reveals the presence of minute particles of a nonmetallic mineral that has retained the luster of its polish. These particles are the small spinel crystals so common in the titaniferous ores. Such an etched section is instructive in showing the relations between the grains of magnetite, ilmenite, and gangue minerals. It has the advantage over thin sections of the ores that it brings out more clearly the relations between the ilmenite and the magnetite grains, and the disadvantage that the recognition of the nature of the gangue minerals is more difficult. Photographs of such sections are shown in Plates II, VIII, and IX.

From these plates it is seen that in most cases the relations between the ilmenite and the magnetite grains are those of the constituents of a granular igneous rock. The texture varies from that of a fine-grained rock to that of a coarse-grained rock. As to the relative abundance of the two minerals, in some ores there is little ilmenite and in others the ilmenite greatly preponderates over the magnetite. The relative size of the grains of the two minerals is of the same order of magnitude, though the ilmenite grains average a little smaller

than those of magnetite. The most common shape of the grains is roughly equidimensional, but they also assume an elongate and, especially in the case of the ilmenite, even a tabloid shape. The effect produced then is as if most of the elongated ilmenite grains followed the outlines of the magnetite grains, with a few distinctly cutting across. This is well illustrated in Plate II, *D*. The tabular grains more frequently terminate bluntly, but some gradually fray out at the ends, as in Plate XI, *A*.

A close megascopic examination of the magnetite areas of many ores will show that much of the magnetite is not a homogeneous mass, but is intergrown with minute particles of ilmenite in both regular and irregular orientation (Pl. II, *C*, and Pl. VIII, *B*). A microscopic examination of sections of such ores reveals the intergrowths to much better advantage, and discloses many intergrowths that are so minute as not to be visible to the naked eye. The nature of these intergrowths is fully discussed under the descriptions of the individual deposits. The results of the study of the intergrowths are for convenience summarized at this point.

In many of the ores there is a tendency for small granules of ilmenite to arrange themselves within a magnetite grain very close to its contact with an ilmenite grain. The granules are usually elongate and lie either with their longer axes parallel to the outline of the ilmenite grain, or at right angles to it. In most cases they form a discontinuous fringe around the ilmenite grains from which they are separated by a narrow zone of magnetite free from inclusions. This condition is shown at the right end of Plate II, *E*. Sometimes the granules coalesce, forming a continuous rim about the large ilmenite grain; and if the elongate particles are oriented at right angles to the outline of the grain, the rim may attain a width of 0.2 mm., as in Plate VI, *B*. Besides the particles that tend to segregate along the contact with larger ilmenite grains, others of irregular to more or less circular shape are scattered here and there through the magnetite (Pl. IV, Pl. VI, *A*, Pl. XV, *B*, and Pl. XVI, *A*). The more common size of these is a maximum dimension of 0.1 to 0.3 mm.

The magnetite grains are at times separated by a thin film of ilmenite, so that in section the outlines are marked by a bright rim of ilmenite as is seen in Plate IV, *A*. Somewhat analogous to this is the crowding of regular intergrowths along definite lines in the magnetite areas, which are often the periphery of the grains (Pl. XIII, *B*, and Pl. XV, *A*).

The most common forms of intergrowths are those that show in the sections as lines and as dots.

The lines are sections of minute ilmenite lamellæ oriented parallel to the octahedral faces of the magnetite. Hence in a simple magnetite crystal they can be oriented in three different planes, and in a

twinned crystal in five different planes. In section, this may give rise to three or to five series of parallel lines; the actual number visible depends of course on the orientation of the section. Plate XV, *A*, illustrates the intergrowths in a simple individual; Plate VI, *A*, the intergrowths in a twinned individual. The angles that the lines make with each other depend also on the orientation of the section. If the section lies parallel to the octahedral face, they intersect at 60°; as the section deviates from this position, the angles approach 0° on the one hand and 90° on the other. But even where the section is so oriented as to expose the three possible directions, intergrowths are not necessarily developed in all three, or at least not equally well developed. Thus, as in Plate III, *B*, the one series of lamellæ is marked by heavy lines, whereas the other two are made up of very small delicate lamellæ. Consequently the pattern produced by the intersecting lamellæ may vary from the equilateral triangular network of a simple magnetite crystal cut parallel to the octahedral face to the more complicated network of five series of intersecting lines in the case of twinned crystals, or to the simplest case of one series of parallel lines, which is most nearly attained in Plate IV, *A* and *B*.

In size the lamellæ range from such as are easily visible with the naked eye (Pl. II, *C*, and Pl. VIII, *B*) to lamellæ so minute as to be scarcely discernible under the highest magnification. The coarsest lamellæ in any of the ores examined occurred in the ores of the Tuscarora mine in Guilford County, N. C. These reach a length of 4 mm. and a thickness of 0.1 mm. They are characterized by local thickenings in the form of protuberances having a thickness as great as 0.2 mm. (Pl. VII, *A*). The distance between the parallel plates averages 0.2 to 0.5 mm., but may be much greater. The plates are distinctly visible to the naked eye and may be peeled off cleavage faces of the magnetite with a knife. Of the other lamellæ examined the only ones approaching the coarseness mentioned were those in the Poplar Lake ore of Minnesota. Aside from these exceptional cases, lamellæ having a length of 1 mm. and a thickness of 0.02 mm. must be regarded as unusually large. Those shown in Plate VI, *A*, and Plate XV, *A*, are among the largest observed.

Generally, instead of one large continuous lamella, there are a number of smaller lamellæ in the same plane, so that in section the effect is that of a broken line instead of a continuous bright line of ilmenite. If unusually thick, they may have a width of 0.02 mm., as in Plate XVI, *A;* but in most cases they are less than 0.005 mm. in thickness, and 0.001 mm. is not at all uncommon. In length every size is represented from lamellæ so minute that their elongation is scarcely discernible to the larger lamellæ mentioned in the preceding paragraph. The delicate short lamellæ showing the broken-line effect are illustrated in Plate IV, *B*, Plate IX, *A*, and in Plate XII, *A*.

As regards the number of lamellæ present, there is also considerable variation. In some ores they are not very numerous and rather widely spaced, as in Plate IV, *A*, Plate XI, *B*, and Plate XVI, *A*. In others they are closely crowded, as in Plate III, *A*, Plate IV, *B*, and Plate VI, *A*. Especially in the case of the more delicate lamellæ the crowding becomes very close, as shown in Plate I, Plate VII, *B*, Plate IX, *A*, and Plate XII, *A*. This crowding becomes in many instances closer and the lamellæ more delicate until the intersecting structure can no longer be discerned, and the magnetite has a peculiar bright luster due to the almost continuous surface of ilmenite. Such grains can easily be distinguished from homogeneous ilmenite grains by the tendency of their luster to show iridescence.

Not only the number but also the distribution of the lamellæ varies greatly. One part of the magnetite surface may be crowded with them so that it shows a closely packed network, and another part, immediately adjacent, may be almost free from intergrowths. Such sudden changes in the distribution of the intergrowths are seen in Plate VII, *B*, and Plate XII, *A*. Then, again, lamellar intergrowths may suddenly terminate and the dots occur in their place as in Plate XI, *B*. Similar to this is the crowding of lamellæ along definite lines in the magnetite, the rest of the area being almost free from lamellæ. These lines are both the outlines between magnetite grains and lines running through the magnetite seemingly with no definite orientation. This effect is illustrated in Plate XIII, *B*, and Plate XV, *A*.

As common as the lamellæ is another variety of intergrowths that show in section as dots with approximately circular outlines. They vary in size from the minutest to the largest, but the impression one receives is that they occur in groups of two sizes—a larger size 0.05 to 0.01 mm. in diameter and a smaller ranging from about 0.002 mm. down to the smallest size observable. Both sizes occur in Plate IV, *B*, and Plate XIII, *A*. Most commonly the dots are irregularly distributed over the megnetite surface, not showing any regular orientation. In some cases, however, they assume a rectilinear orientation parallel to the octahedral cleavages. This is especially prominent with some of the larger dots and is illustrated in Plate XI, *B*, and Plate XIV, *A*. As regards quantity and evenness of distribution the same conditions exist as have been described for the lamellæ. The minute dots may be almost entirely lacking (Pl. XIV, *A*) or they may occur in such numbers as to form almost a continuous surface of ilmenite and produce the peculiar luster on the magnetite that the closely crowded minute lamellæ cause. The uneven distribution of the dots can be seen in Plate XII, *B*, in which there are abundant minute dots at one end and an almost complete absence of them at the other.

One other form of intergrowth has been illustrated in Plate XVI, *B*. This was found in two Canadian ores, the one illustrated from the River Chaloupe and another from a deposit 40 miles north of Montreal. No such intergrowth was observed in any of the ores from the United States. It consists of numerous narrow lenticular bands of what seems to be magnetite, though it is not as readily acted on by the acid as magnetite usually is, intercalated in grains of ilmenite. Hussak [a] found what is probably the same kind of intergrowths in the Brazilian ores from Victoria in Espirito Santo and Caldas in Minas Geraes, the former occurring in a quartz mica diorite, the latter in a basic rock. Of the Victoria ore he says: "The large ilmenite grains show a peculiar lamellation after etching and appear to consist of two substances acted on unequally by the acid, although both are very difficultly soluble in 50 per cent hydrochloric acid." The structure would suggest the solidification of a magnetite-ilmenite eutectic and is worthy of further investigation.

It is at once apparent from the microphotographs that all of the intergrowths that have been described are too minute to admit of a mechanical separation. In many cases they are abundant enough to give the magnetite a considerable titanium content, and herein lies the failure to a large extent of magnetic separation to eliminate the titanium from the ores. But it has also been demonstrated that a magnetite showing none of the intergrowths, but etching with a uniform dull-black surface, may carry a large percentage of titanium. A sample obtained by picking particles of etched magnetite free from intergrowths from sections of Sanford Hill ore contained 6.6 per cent Ti. It would appear, therefore, that the magnetite molecule may itself carry titanium and establish the existence of a mineral "titanomagnetite." The titanium content of titaniferous magnetites that is inseparable by means of magnetic concentration, consequently, occurs both in the form of microscopic inclusions of ilmenite and in the magnetite molecule itself. The quantity of titanium occurring in separable and inseparable form is subject to variation between wide limits. Hence it is impossible to predict beforehand what results the magnetic separation of a given titaniferous ore will yield, but each case must be investigated for itself.

CHEMICAL COMPOSITION OF THE TITANIFEROUS IRON ORES.

In the section of this report dealing with the individual deposits of titaniferous iron ores all the available analyses of titaniferous iron ores have been included. At this point some of the characteristic features of the chemical composition of these ores are discussed. The table following gives the average content of the ores, by districts,

[a] Hussak, E., Über die Mikrostructur einiger brasilianischer Titanmagneteisensteine: Neues. Jahrb. Min., Geol., Paleon., vol. 1, 1904, pp. 99, 108.

in iron, titanium, phosphorus, and sulphur. The figure in parentheses indicates the number of determinations from which the average is made up. The last column gives the amount of titanium per unit of iron.

Average composition of titaniferous magnetites.

Source of ore.	Fe.	Ti.	P.	S.	Titanium per unit of iron.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Iron Mine Hill, R. I.	33.49 (10)	5.85 (10)	0.016 (5)	0.104 (5)	0.1747
New York gabbro ores	44.66 (21)	6.28 (21)	.117 (16)	.237 (11)	.1406
New York anorthosite ores	57.21 (15)	9.61 (15)	.015 (6)	.035 (6)	.1680
New Jersey	57.83 (40)	1.90 (40)	.289 (40)	.3657 (34)	.0329
North Carolina	49.98 (58)	6.61 (56)	.023 (36)	.052 (29)	.1323
Minnesota	49.11 (11)	7.94 (11)	.006 (7)	.126 (7)	.1617
Iron Mountain, Wyo	49.89 (10)	15.96 (10)	Trace.(1)	.87 (3)	.3199
Caribou Hill, Colo	45.57 (4)	12.37 (2)	.020 (2)	.....	.2715
Iron Mountain, Colo	47.86 (11)	7.68 (10)	.026 (8)	Trace. (1)	.1605
Cebolla Creek, Colo	.....	13.61 (2)	.....	.....	.....

One of the most interesting things brought out by this table is the relation existing between iron and titanium. Attempt is made subsequently to show that most if not all of the ores from which the New Jersey averages have been calculated really do not belong to the class of ores under discussion, and hence the New Jersey figures may be ignored. The average titanium content given for the Caribou Hill deposit is beyond doubt wrong. The divergence between Jennings's analysis and Chauvenet's is so great, and the metallographic study of the ore so greatly favors Jennings's determination, that the average of the two certainly does not represent the actual titanium content, but is a figure far above the true value. Hence this average must also be ignored. We then find that the ratio of titanium to iron in the magnetites of all the districts mentioned ranges between the narrow limits of 0.1323 and 0.1747, with the single exception of the magnetite from Iron Mountain, Wyo., which has a ratio twice as great as the average of the others.

The phosphorus content is unusually far below the Bessemer limit. In a few cases, however, the ore contains considerable apatite, which gives it a high phosphorus content. Assumption can not be made, therefore, that a given deposit is low in phosphorus, but it should be tested in this respect. The chances of its exceeding the Bessemer limit are, however, very slight. The sulphur content is also very low, and in no case high enough to produce any deleterious effects.

The geology, origin, and composition of the titaniferous iron ores have been discussed at considerable length by Vogt in a series of articles that appeared in the "Zeitschrift für praktische Geologie" during the years 1893, 1894, 1900, 1901, and 1910. A brief summary of

these results is given in "Die Lagerstätten der nutzbaren Mineralien und Gesteine,"[a] by Beyschlag, Krusch, and Vogt.

Vogt finds that the iron-ore segregations occurring in gabbros, labradorfels, and augite and nepheline syenites are characterized all over the world, without exception, by a considerable titanium content. The titanium in the relatively rich segregations rarely falls below 4 to 5 per cent. Another element that is rarely lacking is vanadium in the form of V_2O_3, and there may also be present Cr_2O_3, NiO, and CoO. The manganese content is usually low. The apatite or P_2O_5 content is generally low. The quantity of sulphide is usually small, and in a few cases carbon in the form of graphite occurs. A review of the analyses of the titaniferous ores of the United States showed that they conform in every respect to the characteristics outlined by Vogt.

As regards their manner of occurrence, Vogt states that they occur in the central parts of the eruptive masses into which they gradate as irregular schlieren, or as distinct dikes with sharp contacts against adjacent eruptive rock. All these types of occurrences are represented among the deposits of the United States. The manner in which the differentiation progresses is discussed by Vogt and is illustrated graphically by diagrams, and the theory of the processes of differentiation is considered. A consideration of these points is, however, beyond the scope of this report.

Of more immediate interest is Vogt's discussion of the relations of the iron, titanium, phosphorus, and sulphur of the ores to the original content of these elements in the magma. Omitting certain analyses that he regarded as unreliable, the analytical results collected by him covering occurrences all over the world showed that the ratio between titanium and iron ranges between 0.063 and 1. Only in rare cases, however, does the ratio fall without the limits 0.1 and 0.667; and in the vast majority of cases of segregations within gabbros and anorthosites relatively rich in iron-magnesian silicates the ratio lies between 0.111 and 0.250. The table on page 35 shows that all of the American occurrences, ruling out those of New Jersey and Caribou Hill for reasons already stated, fall within this narrow limit with the exception of the Iron Mountain, Wyo., deposit. This deposit occurs in anorthosite relatively poor in ferromagnesian minerals, and such deposits Vogt states are characterized by a higher titanium content. A thoroughly trustworthy ratio of titanium to iron in the original gabbroitic magma can not be given, but he regards it as falling below 0.1 to 0.125. Hence in the processes of differentiation not only has there been an absolute enrichment of iron and titanium, but titanium has to a varying extent been enriched relatively to the iron.

[a] Vol. 1, 1909, pp. 247–253.

According to Clarke,[a] the eruptive rocks carry an average of 0.11 per cent of phosphorus and 0.11 per cent of sulphur; but the gabbroitic rocks carry more than the average quantity of these two elements, and Vogt assumes as the minimum an average of 0.2 per cent of phosphorus and 0.15 per cent of sulphur for these rocks. The average phosphorus content of the titaniferous iron ores is far below this figure, and in only a few cases does it reach that quantity in individual deposits. The high-grade ores usually carry only the minutest quantity of phosphorus. The sulphur content is also usually very minute. In a few cases there is a considerable sulphur content in the ores owing to the presence of sulphides, but in most cases the average is below the average for the magma. Consequently, the assumption is warranted that while the iron and titanium were being enriched to form the ore bodies the phosphorus and sulphur remained in the magma. In other cases there have been segregations of apatite and of sulphidic ores. Vogt concludes, therefore, that the conditions favorable to the segregation of iron and titanium differ from those favorable to the segregation of phosphorus, and these again from those favorable to the segregation of sulphides; and that in most cases these conditions do not exist at the same time or place.

He also takes up the behavior of some of the less essential constituents of the ores in this respect. A slight enrichment of manganese has taken place, but not to the extent to which iron has been enriched. Kemp found that chromium and vanadium vary in the same manner if not absolutely in the same degree. This relation is corroborated and the average enrichment of these two elements in the ores given as tenfold. The nickel and cobalt content is usually small and in most cases less than one would expect.

In conclusion, it may be stated that the characteristics of the chemical composition of the titaniferous iron ores of the United States are the same as those that mark these ores the world over. As regards iron content most of them must be classed as medium to low grade ores. Fortunately, however, the deposits with the greatest economic possibilities are the richest and their ores are high grade. Phosphorus and sulphur are present in objectionable quantities in only rare instances; and phosphorus especially is unusually low in the most promising deposits. The slag forming compounds, lime, magnesia, alumina, and silica, are present almost entirely in the gangue minerals, and play no important rôle in the composition of the ores. The high-grade ores in which the iron content is high enough for direct furnace use contain little gangue. The lower-grade ores would have to be subjected to magnetic concentration to bring their iron content within the required limits, and this would reduce

[a] Clarke, F. W., The data of geochemistry: U. S. Geol. Survey Bull. 491, 1911.

the percentage of those constituents in the concentrates to a minimum.

ECONOMIC IMPORTANCE OF THE DEPOSITS.

The field investigation of the titaniferous iron ores has given the author the impression that the possibilities of these ores with reference to the iron-ore resources of the country have been grossly exaggerated in many quarters. In the detailed description of the more important deposits of these ores that follows, their commercial aspect is discussed so far as available data make this possible, and the statements made here are in the nature of conclusions based on those discussions.

Least promising are the possibilities of the ores in the ferromagnesian rich rocks. As a general rule, the deposits are of small and irregular extent, and the ores fine grained and lean. Leaving out of consideration for the moment their titanium content, they are ores that would require magnetic concentration before they could be used in the blast furnace. Further, most of the deposits are inaccessible to adequate transportation facilities. Their situation both topographically and with regard to the industrial development of the country at large is such that there are no prospects of transportation facilities being provided for a long time to come. Hence, to put these deposits on a producing basis, the heavy initial expense would have to be incurred of erecting magnetic concentrators and providing adequate transportation facilities. The small extent of most of the deposits and their irregular distribution make those costs prohibitive. Then, their titaniferous character must be taken into consideration. The relations of ilmenite and magnetite in the ores are such that the least favorable results would be obtained in the elimination of titanium by magnetic concentration. In fact, the titanium would be increased together with the iron; so that the higher the iron content of the concentrates, the higher would be their titanium content.

In the case of the ores in the anorthosites and the feldspar-rich gabbros, the conditions are much more favorable. As a general rule the ores are much coarser grained and richer. This tendency to coarser-grained, rich ores is noticed where the gabbros assume an anorthositic facies. Thus in some of the feldspathic phases of the Duluth gabbro, in Minnesota, the ores are coarser grained than the normal gabbro ores, and small segregations of comparatively pure ore occur. Also at Iron Mountain, Colo., the immediate country rock shows an anorthosite phase, and the ores are richer and coarser grained than ordinary gabbro ores and greatly resemble the typical ores in anorthosite.

Two deposits in anorthosite stand out above all other titaniferous iron-ore deposits in the United States; these are at Sanford Hill, N. Y., and Iron Mountain, Wyo. At each of these localities, in addition to the one enormous deposit, there are smaller but important deposits with the same kind of ore. At neither place has a careful estimate of available tonnage been made, but it is apparent from the known extent of the deposits that the quantity of ore that can be mined by simple quarrying methods alone runs into millions of tons. The ores are nearly pure aggregates of magnetite and ilmenite, the gangue minerals being so subordinate in amount as to be negligible. The average composition of the Adirondack ore corresponds to 62.8 per cent magnetite and 30 per cent ilmenite, whereas that of the Wyoming ore is equivalent to 49.8 per cent magnetite and 40.15 per cent ilmenite. The results of magnetic concentration tests show that the Adirondack ore would yield a concentrate containing less than 7 per cent Ti and the Wyoming ore a little less than 10 per cent. The titanium content of the Adirondack concentrate is low enough to make feasible its mixture with several parts of nontitaniferous ore to make a product low enough in titanium to be used in ordinary blast-furnace practice. The low phosphorus and sulphur content of the concentrate would make it a desirable material for mixing with ores of non-Bessemer grade and high in sulphur. The concentrates of the Wyoming ore are rather high in titanium for this purpose, and it is probable that these deposits will have to await some such process as electrical smelting that will make possible the direct smelting of titaniferous ores. To provide adequate transportation facilities for the Sanford deposit will require the construction of more than 30 miles of railroad over a mountainous country; for the Iron Mountain deposit, only 10 miles will be necessary, and the construction will be much easier. In both cases the deposits are large enough to warrant this expense.

To sum up, it may be confidently asserted that the Lake Sanford and Iron Mountain, Wyo., deposits are important ore bodies that before many years will be furnishing a considerable tonnage of iron ore. The rest of the titaniferous iron-ore deposits of the United States can not for many years to come be of any value as sources of iron ore.

CHARACTER, SITUATION, AND POSSIBILITIES OF THE DEPOSITS.

In the subsequent pages the more important deposits of titaniferous iron ores in the United States are described individually. The order of treatment is geographic, beginning in the northeast with Rhode Island and continuing southward and westward. The method

pursued has been to give first a summary of the geologic features of a given deposit in order to prepare the way for an adequate discussion of the metallographic features of its ores and of its economic possibilities.

RHODE ISLAND

IRON MINE HILL DEPOSIT.

The titaniferous magnetic iron ore of Iron Mine Hill, R. I., has been known for over two centuries, and has been of such interest to geologists that repeated mention of the deposit is made in geological literature. The most recent and thorough study of this deposit is that by Johnson and Warren,[a] who made a careful study of the deposit itself, and included in the presentation of their results a historical account of the deposit and a summary bibliography of the pertinent literature. In view of this thorough study and careful description of the deposit, the author of this report has not deemed it necessary to revisit the locality. The following description of the deposit being based chiefly on the paper by Johnson and Warren.

LOCATION.

Iron Mine Hill lies in the northeast corner of Rhode Island, near the Massachusetts line, in the town of Cumberland, about 3 miles east of Woonsocket. The region is heavily drift covered and of moderate relief, the hills not rising much more than 100 feet above the surrounding country. One of these hills consists of a heavy rock which early attracted attention as a possible source of iron ore, and it has since received the name Iron Mine Hill. The dimensions of the hill as given by Jackson[b] are 462 feet long, 132 feet broad, and 104 feet high. The area occupied by the ore body is even larger. The exact boundaries are covered, but it appears from surface indications and magnetic observations that the ore body has a length of 1,200 feet and a width of 500 to 600 feet. The direction of the longer dimension is nearly north and south.

HISTORY.

Repeated attempts have been made to utilize this ore. As early as 1703 it was mixed with hematite from Cranston, R. I., and used in a foundry probably located in the town of Cumberland. Part of the cannon used against Louisburg in 1745 were made at this forge, and the ores were utilized for this purpose during the Revolutionary War. Attempts were made to use the ores during the early and middle parts of the last century by mixing them with high-grade

[a] Contributions to the geology of Rhode Island. I, Notes on the history and geology of Iron Mine Hill, Cumberland, by B. L. Johnson. II, The petrography and mineralogy of Iron Mine Hill, Cumberland, by C. H. Warren: Am. Jour. Sci., 4th ser., vol. 25, January, 1908, pp. 1–38.

[b] Jackson, C. T., Report on the geological and agricultural survey of the State of Rhode Island, 1840, p. 82.

ores at furnaces in New York, Pennsylvania, and New England. In all several thousand tons of ore have been used. The great drawback has always been their low iron content and the high percentage of titanium that they carry. The enormous quantity of the ore and the low phosphorus and sulphur content have been advantages that have partly offset the disadvantages. However, no attempt to work the ores for their iron has been made for many years, and an attempt to use the rock for road metal (macadam) at the close of the last century also met with failure, probably because its extreme toughness made its quarrying too expensive.

ORIGIN.

Jackson[a] was the first to recognize the eruptive character of the ore. The first microscopic study of the ore was made in 1881 by Wadsworth,[b] who called attention to its similarity to the celebrated iron ore of Taberg, Sweden. Though the origin of the rock could not be told from its field relation, he considered that its mineral association and microscopic characters indicated that it was an eruptive rock. He therefore concluded that the deposit was of eruptive origin. In 1884, Wadsworth[c] proposed the name cumberlandite for this and similar rocks.

Opposed to the eruptive theory at this time were men like Hitchcock[d] and Dana,[e] who considered the magnetite to be of metamorphic origin.

At present the igneous origin of the cumberlandite is not doubted, the general opinion being it is probably a segregation from a gabbroitic magma.

GENERAL FIELD RELATIONS.

Unaltered cumberlandite is exposed at only one point on the western slope of the hill. This area of fresh rock, about 300 to 400 square feet, passes over, with a comparatively narrow transition zone, into the altered types that form the greater part of the hill. One of the first changes in the rock consists in the alteration of the feldspar phenocrysts to dark-green chloritic aggregates, in association with which a little actinolite may occur. Such rock makes up the greater part of the exposures. As alteration progresses the olivine also passes into chlorite and actinolite, and in the final stage olivine and actinolite are changed to serpentine. The analysis cited below, from Wadsworth, was probably that of a sample of ore in which the feldspar had undergone changes.

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[a] Op. cit., p. 53.

[b] Wadsworth, M. E., A microscopic study of the iron ore, or peridotite, of Iron Mine Hill, Cumberland, R. I.: Mus. Comp. Zool. Harvard College Bull. 7, 1881, pp. 183–187.

[c] Wadsworth, M. E., Lithological studies: Mem. Mus. Comp. Zool. Harvard College, vol. 11, 1884 p. 80.

[d] Hitchcock, Edward, Final report on the geology of Massachusetts, 1841, p. 554.

[e] Dana, J. D., Iron ore of Iron Mine Hill, Cumberland, R. I.: Am. Jour. Sci., 3d ser., vol. 22, 1881, p. 152.

The rock is cut by a major system of joints that is parallel to the long direction of the hill and has a nearly vertical dip. A minor series of joints, approximately at right angles to these, cuts the hill into small, rudely prismatic blocks. Many of these blocks have been plucked out by the ice, leaving an irregular broken surface. Shearing movements which have produced a schistose structure in the secondary actinolite and chlorite formed along the joint planes.

The cumberlandite is nowhere exposed in contact with any other rock, and is separated from the nearest outcrops by barren areas for a distance of at least 700 feet. The surrounding rocks consist of green chlorite-epidote schist and quartzose biotite schist cut by numerous quartz veins and granite dikes, granite, diabase, and a coarse-grained metamorphosed gabbro. The gabbro extends in a series of heavily glaciated ridges around the west side of the hill for a distance of three-quarters of a mile, obtaining a width of one-quarter of a mile. It is everywhere separated from the cumberlandite by swamps and drift, so that no contact can be observed.

The occurrence of inclusions of gabbro in the cumberlandite at several places indicates a not distant contact. Moreover, a line connecting the known northeastern and southeastern limits of the gabbro would pass close to the supposed western limit of the cumberlandite. The evidence, therefore, indicates the intrusion of the cumberlandite into the gabbro. The basic character of both, their titanium content, and their occurrence as adjoining masses surrounded by genetically unrelated rocks suggested to Johnson that they are offshoots from a common magma. Such a differentiation within the magma and subsequent intrusion of the one differentiate into the other instead of differentiation in situ has, according to the observations of the author, very frequently occurred in the case of the titaniferous magnetites. Though this deposit has many features in common with the well-known Taberg deposit in Sweden, resemblances that are just as strikingly brought out in the metallographic study of the two ores, this is one essential difference in the two deposits. The Taberg ore is an example of differentiation in situ. The ore, a magnetite olivenite, has segregated in the center of a mass of *olivine* hyperite. Here there is a transition from country rock to ore; at Iron Mine Hill, the ore was intruded into the gabbro, and if exposed the contact would be sharp and not gradational.

The gabbro is not cut by the surrounding granite, and no inclusions of the granite or sedimentaries in the gabbro are known. Positive evidence of the relative ages of the gabbro and the surrounding rocks is lacking. Johnson[a] suggests that the gabbro is younger than the schists but older than the granite, since such a sequence has been determined in the case of a mass of gabbro surrounded by schist

[a] Johnson, B. L., and Warren, C. H., Contributions to the geology of Rhode Island: Am. Jour. Sci., 4th ser., vol. 25, January, 1908, pp. 10–11.

and granite at Norwich, Conn., about 80 miles southwest of Cumberland.

The gabbro was originally a rather coarse-grained rock, consisting of labradorite, averaging about Ab_1An_1, a purplish augite of distinctly diabasic habit, an unusual proportion of ilmenite in the form of large grains closely associated with the augite, and accessory apatite. Shearing and metamorphism have affected the gabbro mass unevenly. The central and southern parts are least affected; the northern parts have been more severely altered, and in part changed into fissile green schists.

CUMBERLANDITE.

The cumberlandite has a characteristic appearance and can be readily distinguished macroscopically from all other known titaniferous magnetites in the United States. In a grayish-black, fine-grained groundmass of irregular fracture are embedded relatively large tabular crystals of plagioclase. Two constituents of the groundmass can be recognized with the eye alone. One is a yellowish glassy olivine, the other a grayish-black mass of magnetite and ilmenite.

Plagioclase makes up about 13 per cent of the rock. The crystals average 1 cm. square by 2 mm. thick and range in size from minute grains to a maximum of 2 cm. square by 2 to 3 mm. thick. The common segregation of two or more feldspar crystals gives the rock a cumulophyric texture. Locally a tendency to parallel or fluidal arrangement is noticeable, but there is no persistent direction of orientation maintained. Though the feldspar is more abundant in one place than in another, it has a fairly uniform distribution throughout the rock.

At the contacts between the feldspar and the groundmass reaction rims are always developed; but they are exceedingly narrow, ranging from 0.003 to 0.01 mm. At the feldspar-olivine contacts, the reaction rim consists of pale-green to colorless needles of actinolite that are perpendicular to the contact. At many ilmenite-magnetite-feldspar contacts shreds of biotite appear; and in some places they occur as a narrow border next to the magnetite, with a border of actinolite on the feldspar side.

The texture of the groundmass is well shown on a polished surface of the ore, especially after etching with strong hydrochloric acid, which causes the dark glassy olivine to lose its high luster and to stand out as a light-yellow lusterless substance in a network of magnetite and ilmenite. The olivine occupies somewhat more than half of the surface.

In thin sections the olivine is seen to consist of a mosaic of anhedral grains ranging in size from minute particles to individuals 5 mm. long by 2.5 mm. wide. In general the olivine lacks crystallographic outlines, yet throughout the rock the magnetite-ilmenite

acts as the matrix. The following analysis, made by Warren, shows it to be a hyalosiderite:

Results of an analysis of olivine from cumberlandite.

	Per cent.
SiO_2	37.16
TiO_2	.07
Fe_2O_3	.12
FeO	31.38
MnO	.40
CaO	Trace.
MgO	31.16
Feldspar (insoluble)	.34
	100.63
Fe	24.5

Spinel occurs at random in the ore in the form of minute subangular grains, reaching a maximum size of 0.45 mm. by 0.12 mm. More commonly they are equidimensional grains averaging 0.10 to 0.20 mm. in diameter. The spinel is seen in thin sections as a dark-green, feebly transparent isotropic mineral, which was called glass by Wadsworth, but thought to be hercynite by G. H. Williams. Warren calls it pleonaste.

CHEMICAL COMPOSITION OF ORE.

The results of a number of analyses of cumberlandite are available, but the only complete one is that made by Warren. The available results are presented below:

Results of analyses of Iron Mine Hill ore.

Constituent.	1 [a]	2 [b]	3 [c]	4 [d]	5 [e]	6 [f]	7 [g]	8 [h]	9 [i]	10 [j]
SiO_2	22.35	23.00	22.87	20.85	20.85	20.77	26.33	.....	.....	20.41
Al_2O_3	5.26	13.10	10.64	5.55	5.26	4.01	.....	.....	.....	.....
Fe_2O_3	14.05	27.60	44.88	45.62	16.32	17.22	58.50	.....	.....	.....
FeO	28.84	12.40			28.53	26.51		.....	.....	.....
TiO_2 [k]	10.11	15.30	9.99	9.98	9.63	10.08	3.66	9.35	9.86	11.75
MnO	.43	2.00	2.05	.....	.....	.....	2.10	.....	.....	.05
CaO	1.17	.....	.65	.73	.98	.53	.65	.....	.....	.....
MgO	16.10	4.00	5.67	16.45	17.60	17.96	6.80	.....	.....	.....
Zn	.71	.....	.20	.....	.....	.....	.....	.....	.....	.....
H_2O	.42	2.60	3.05	.....	.66	2.70	1.96	.....	.....	.....
P_2O_5	.02	.....	.....	Trace.	.....	.....	.....	.026	.026	.103
S	.38	.....	.....	Trace.	.....	.....	.....	.057	.057	.023
V_2O_3	.18	.....	.....	.....	.....	.....	.....	.....	.....	None.
Cr_2O_3	Trace.	.....	.....	.....	.....	.....	.....	.....	.....	.....
Na_2O	.44	.....	.....	.....	.....	.....	.....	.....	.....	.....
K_2O	.10	.....	.....	.....	.....	.....	.....	.....	.....	.....
CO_2	.02	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cu	.08	.....	.....	.....	.....	.....	.....	.....	.....	.....
Co-Ni	.08	.....	.....	.....	.....	.....	.....	.....	.....	.....
Pb	Trace.	.....	.....	.....	.....	.....	.....	.....	.....	.....
Total	100.74	100.00	100.00	99.13	99.83	99.78	100.00	.....	.....	.....
Fe [l]	32.5	28.96	32.49	33.03	33.67	32.73	42.85	32.80	32.84	33.87

[a] Warren, C. H., Contributions to the geology of Rhode Island. The petrography and mineralogy of Iron Mine Hill, Cumberland: Am. Jour. Sci., 4th ser., vol. 25, January, 1908, p. 24.
[b] Jackson, C. T., Geol. Survey of R. I., op. cit., p. 53.
[c] Thurston, R. H., quoted by Wadsworth: Mus. Comp. Zool. Harvard College Bull. 7, 1881, p. 185.
[d] Drown, T. M., quoted by Wadsworth: pp. 16–17.
[e] Packard, R. L., quoted by Wadsworth, M. E.: Mem. Mus. Comp. Zool. Harvard College, vol. 11, pt 1, 1884, Report of the State Board (of Michigan) Geol. Survey for the years 1891–92, Lansing, 1893, p. 96 (ore with feldspar).
[f] Same as foregoing (ore without feldspar).
[g] Chilton, —, quoted by Holley, A. L.: Trans. Am. Inst. Min. Eng., vol. 6, 1878, p. 226.
[h] Tenth Census U. S., vol. 15, 1886, p. 567.
[i] Same as foregoing, but dried.
[j] Data furnished by W. L. Cummings, of the Bethlehem Steel Co.
[k] Average, 9.75.
[l] Average, 33.49.

Calculating its position in the quantitative classification, Warren found that the ore fell in an order and rank for which there was no name, and at the suggestion of Prof. L. V. Pirsson he proposed the names rhodare and rhodose from the State in which the rock occurs. The stem cumberland was rejected in order to prevent confusion with the better known town of Cumberland in England. The mode as calculated by Warren is given below.

Mode of cumberlandite.

Mineral.	Percentage by weight.	Percentage by volume.
Orthoclase	0.56	0.8
Labradorite (Ab_3An_4)	9.23	13.7
Olivine	46.08	49.4
Magnetite	20.65	15.9
Ilmenite	18.68	15.2
Spinel	3.55	3.9
Sulphides	1.15	1.1

Because the name cumberlandite is widely known its use in this discussion has been deemed advisable.

METALLOGRAPHIC DESCRIPTION OF THE ORE.

As already stated, the ore minerals form the matrix for the olivine and the other minerals of the groundmass. They occur in irregular areas of about the same size as the olivine grains or a little smaller, and except in the altered phases are more or less connected with each other. They form a little less than 50 per cent of the groundmass.

When a polished surface of the ore is etched with hydrochloric acid a part of the ore remains bright, whereas the greater part shows a dull-black background on which a delicate reticulate structure has been developed. The bright grains, which are ilmenite, are usually quite small, rarely having a maximum dimension of 2 mm., and form only a small percentage of the ore. The reticulated structure of the ilmenite growth in the magnetite is especially characteristic in this ore. It is marked by the minuteness of the lines and the closeness with which they are crowded. They range in fineness from such as are shown in Plate I, *A*, down to such minute intergrowths as not to be resolvable by the magnification used in the figure. Such areas are distinguishable under the metallographic microscope from areas of true ilmenite by their peculiar iridescent luster. The coarsest of the lines attain a length of only 0.15 mm. The same delicate minute network is also characteristic of the Taberg ore with which the Iron Mine Hill ore has so often been compared geologically. The similarity of the structure is brought out in Plate I, *A* and *B*.

The metallographic features of this ore distinguish it from all the other American ores that were studied, as is at once apparent from a comparison of the microphotographs shown in this report.

POSSIBILITIES OF UTILIZATION.

The size of the deposit and its low phosphorus and sulphur content would indicate the desirability of working it. However, more than offsetting these advantages are its low iron content and its titaniferous character. The low percentage of iron could be overcome by magnetic separation were not the character of the ore such as to make magnetic concentration impracticable. The magnetite and the ilmenite are so intimately intergrown that with the exception of the larger grains of ilmenite the ilmenite would be concentrated equally with the magnetite.

The result of a perfect separation of the ilmenite and the magnetite from the gangue minerals can be calculated from the mode of the rock given above. About one-third of the iron of cumberlandite is contained in the olivine, and this would go into the tailings, leaving only two-thirds or about 22 per cent Fe for the concentrates. Again, assuming that one-third of the titanium could be removed by the separation of the larger grains of ilmenite, a percentage greater than is actually the case, the iron would be still further decreased by more than 2 per cent and the titanium by nearly 2 per cent. Such a perfect concentrate would contain, therefore, 20 per cent of cumberlandite in the form of iron and 4 per cent in the form of titanium, in a bulk one-third by weight of the original rock. In other words, it would take 3 tons of ore to produce 1 ton of concentrates analyzing 60 per cent Fe and 12 per cent Ti.

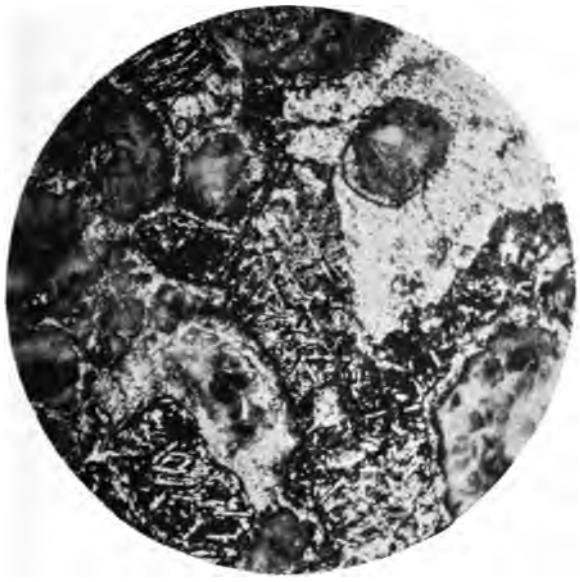
A concentration made on a commercial basis would show results still less favorable. The percentage of iron would fall below 60 per cent without a corresponding decrease in titanium, as, with a less complete separation, a smaller percentage of ilmenite than that assumed above would be removed. Consequently the cost of obtaining the degree of concentration necessary to produce a product high in iron would be excessive, even if the resulting product would not be unsalable on account of its high titanium content.

NEW YORK.

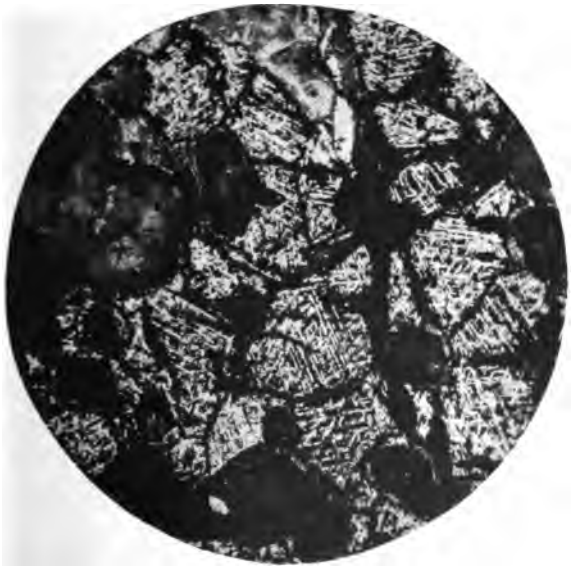
Titaniferous iron ores occur in New York State in the Adirondack Mountains and on the east bank of the Hudson River below Peekskill.

DEPOSITS IN THE PEEKSKILL REGION.

At this locality there is an area of about 25 square miles that consists of rocks of the gabbro family known as the Cortlandt series. Associated with these rocks are highly aluminous titaniferous ores



A. IRON MINE HILL, R. I., ORE. (X 60.)



B. TABERG, SWEDEN, ORE. (X 60.)

which have attracted considerable attention both as a source of iron and as a source of emery. These ores differ from normal titaniferous magnetites in their low titanium content and in the high percentage of alumina that they carry. The percentage of TiO_2 ranges from 0.15 to 4.15 per cent, and that of alumina from 20 to nearly 50 per cent. Mineralogically they consist of spinel, magnetite, and corundum. The percentage of iron ranges from about 30 to 40 per cent, but averages about 33 per cent. The deposits do not seem to be of any importance from the standpoint of iron ores, and their titanium content is so low as not to seriously interfere with their use in ordinary blast-furnace practice, especially if mixed with nontitaniferous ores. Hence no special study of them was made in connection with this report. They have been described by Dana [a] and by Williams.[b]

DEPOSITS IN THE ADIRONDACK REGION.

The titaniferous magnetites of the Adirondack region belong to the normal type. The ores have been studied in considerable detail by Kemp, and in his papers are more fully described than any other deposits of titaniferous ores in the United States. The most complete account is that on "The Titaniferous Iron Ores of the Adirondacks."[c] More recently Newland [d] published an account of these ores, which, however, he says is based chiefly on the descriptions and conclusions of Kemp. The deposits in the gabbros in the townships of Elizabethtown and Westport were again described in 1910 by Kemp, some additions being made to the earlier descriptions.[e] These articles refer to previous contributions and include the information available at the time they were written. The following descriptions of the occurrences are based chiefly on Kemp's accounts, and are supplemented by additional information taken from Newland's bulletin and observations made by the author at those deposits visited by him.

DISTRIBUTION OF THE ORE DEPOSITS.

The titaniferous ores of the Adirondacks are associated in all but one doubtful case with rocks of the gabbro family. Consequently their distribution is determined by that of the gabbro rocks. The principal area of these rocks is in Essex and southern Franklin Counties, where they occupy about 1,200 square miles. Smaller outlying intrusions are found in Clinton and Warren Counties and in the

[a] Dana, G. D., Geological relations of the limestone belts of Westchester County, N. Y.: Am. Jour. Sci., 3d ser., vol. 20, 1880, pp. 199-200.

[b] Williams, G. H., The norites of the "Cortlandt series" on the Hudson River near Peekskill, N. Y.: Am. Jour. Sci., 3d ser., vol. 33, 1887, pp. 194-199.

[c] Kemp, J. F., 19th Ann. Rept. U. S. Geol. Survey, pt. 3, 1899, pp. 377-422.

[d] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 3, 1908, pp. 146-170.

[e] Kemp, J. F., and Ruedemann, R., Geology of the Elizabethtown and Port Henry quadrangles: N. Y. State Mus. Bull. 138, 1910, pp. 137-149.

western Adirondacks. Most of the deposits occur in the townships of Westport, Elizabethtown, and Newcomb, in Essex County.

FIELD RELATIONS OF THE ORES.

The principal geological features of the deposits are concisely stated by Kemp [a] and his account is quoted nearly in full in the paragraphs following:

The titaniferous ores of the Adirondacks are associated in all cases [b] with rocks of the gabbro family and with two well-marked varieties belonging to it. The two are contrasted in appearance and are easily recognized. The largest ore bodies are contained in rocks that are chiefly labradorite, in the coarsely crystalline granitoid aggregates that have been called anorthosite by the Canadian geologists, within whose territories they are abundant. A little augite or hypersthene is always present in a very subordinate capacity, and rims of garnet are common around the bisilicates. The ores in anorthosites are limited to Newcomb, Wilmington, and North Hudson townships, and lie among the highest peaks of the mountains. In their geological associations and general character they are practically like the huge masses of ore that occur along the lower St. Lawrence in Quebec and near Ekersund in southwestern Norway.

The smaller ore bodies, although still of very considerable size, are found in a dark basic gabbro or norite, in which the ferromagnesian silicates predominate over the feldspar. Labradorite is the feldspar and green augite the chief bisilicate. Hypersthene is sometimes abundant and olivine is frequently present. Small garnets in granular rims about the bisilicates are quite invariably present, and shreds of biotite are by no means uncommon. Brown hornblende is an important constituent in and near the ore, and is also common throughout the mass of the rock. Small, irregular anhedra of titaniferous magnetite are always disseminated in the gabbro, and apatite is, of course, not lacking. In texture the rock approximates the diabasic granular type, because of the marked tendency of the feldspar to form lath-shaped individuals.

The ores in the gabbro are contrasted in physical appearance with those in the anorthosite. The former are closely crystalline and dark gray, with a dense, compact aspect, like a finely crystalline igneous rock. The titaniferous magnetite is in relatively small individuals and is mingled with ferromagnesian silicates. The ore from the anorthosites is coarsely crystalline, like ordinary magnetites. It sometimes has a bluish shade, which is regarded by many as characteristic. It lacks the abundant ferromagnesian silicates, and contains large feldspars instead. The greater coarseness of crystallization is in great part due to the larger masses in which it occurs. The contrasts in the grain of the ores, however, are fully reproduced in the textures of the wall rocks, and the ores vary with the varying conditions under which each has crystallized.

The anorthosites and gabbros belong to a great series of intrusives which have come up through older gneisses and crystalline limestones. Considered in their entirety, the intrusives embrace other varieties of gabbroic rocks than the two just cited, varieties that lie between them as extremes, but as the intermediate types are not immediately concerned with the ores they are not described in further detail. The

[a] Kemp, J. F., The titaniferous iron ores of the Adirondacks: 19th Ann. Rept. U. S. Geol. Survey, pt. 3, 1899, pp. 397-399.

[b] A single exception is the Port Leyden mine in Lewis County, outside of the main gabbro area. The immediate wall rock of the deposit is not exposed, but it seems to be associated with quartz gneisses containing potash feldspar. The anomalous relations of this deposit are probably only apparent and it seems reasonable that, as Newland suggests, "it may be related to some underlying magma from which the ore body represents an offshot, perhaps intrusive in the granitic gneiss." One should hesitate to consider a deposit whose relations to its wall rock are as obscure as in this case an exception to a rule of such general application.

intrusives correspond to the Norian of the Upper Laurentian of the Canadian geologists, and the gneisses and crystalline limestones are the representatives of the Canadian Grenville series. In the nomenclature adopted by the United States Geological Survey, both the older rocks and the intrusives must be included in the Algonkian on account of the presence of the limestones. After the intrusions of the gabbros and anorthosites the Adirondack region was subjected to great dynamic metamorphism, so that the massive rocks were crushed and often drawn out into gneisses, and so involved with the older formations as to greatly obscure the stratigraphical relations. It is rather rare to find one of the Norian massive rocks that does not display some evidence of this metamorphism, but in a few cases limited areas of the dark gabbros have escaped it to a greater or less degree, and in the western part of Essex County the anorthosites are not greatly affected.

DESCRIPTION OF OCCURRENCES IN ELIZABETHTOWN, WESTPORT, AND MORIAH TOWNSHIPS.

Only those occurrences that were visited by the author are described. They include nearly all the more important deposits and are sufficiently characteristic to establish clearly the nature of the ores.

SPLIT ROCK MINE.

The Split Rock mine is located on the west shore of Lake Champlain in a small cove, a half mile above Snake Den Harbor and 5 miles northeast of Westport. It is on a steep hillside overlooking the lake. The hill is a spur of Split Rock Mountain which is a long ridge running parallel to the lake shore. Split Rock Mountain consists chiefly of anorthosite with some gabbro. The wall rock of the ore is a highly garnetiferous gabbro. The garnet occurs throughout the rock in small clusters of nearly black crystals, giving the rock a peculiar mottled appearance. The plagioclase of the gabbro consists of nuclei swarming with minute inclusions of pyroxene and probably spinel, surrounded by areas of clear feldspar. The garnet tends to occur in the areas of clear feldspar and closely associated with the femic minerals. Cutting the rock and the ore are small dikes of a white rock varying from a few inches to a foot in thickness. In thin section this rock is seen to consist almost entirely of saussurite with a few scattered bunches of augite, garnet, and hornblende.

Three ore bodies outcrop within a distance of 200 feet. The larger of these, which is the more southerly, has a width of about 10 feet. Two openings have been made on this ore body about 20 feet apart. An opening has also been made in the most northerly of the three deposits.

These ore bodies are magnetite-rich lenses into which they grade by gradual transition. Most of the ore is a dark-colored fine-grained rock of seeming good quality, but polished sections show that it is a very lean ore. Plate II, *A*, shows a specimen of the ore that has been polished and then leached with concentrated hydrochloric acid until the gangue minerals are all light colored. The relations and

relative quantities of ore minerals and gangue are well brought out. A small amount of coarser-grained ore occurs and this carries little gangue. A peculiar feature of this ore and the ore from Little Pond, which has been described by Kemp, is the occurrence through it, in veinlets and in crusts coating the bounding surfaces of cracks, of a green isotropic material which is presumably a basic glass; the relations of the glass to the ore are such as to indicate a primary origin that is a most anomalous phenomenon in a plutonic rock. The following results of analyses of wall rock and ore are taken from Kemp:

Results of analyses of Split Rock gabbro and ore.

Constituent.	1[a]	2[b]	3[c]
SiO_2	47.88	17.60	16.46
TiO_2	1.20	15.66	14.70
Cr_2O_3	Trace (?)	.51	.....
Al_2O_3	18.90	10.23	.34
Fe_2O_3	1.39	15.85	38.43
FeO	10.45	27.94	23.40
NiO.CoO	.02	[d]	[d]
MnO	.16	Little.	.23
CaO	8.36	2.86	3.54
SrO	Trace.	.....	.....
BaO	Trace.	.....	.....
MgO	7.10	6.04	2.13
K_2O	.81	[d]	.....
Na_2O	2.75	[d]	.....
Li_2O	[d]	.....	.....
H_2O below 110°	.18	1.33	.....
H_2O above 110°	.43		.....
P_2O_5	.20	.04	.....
V_2O_3	Trace.	.55	.....
CO_2	.12	.10	.....
S	.07	.14	.....
	100.02	99.15	.....
Fe	.....	32.82	32.59
Specific gravity	3.090	4.138	.....

[a] Wall rock; analyst, W. F. Hillebrand.
[b] Ore; analyst, W. F. Hillebrand.
[c] Ore; analyst, G. W. Maynard.
[d] Not determined.
[e] Faint trace.

Analyses 1 and 2 have been recast by Prof. Kemp. Analysis 3 could not be recast on account of the great excess of Fe_2O_3 over FeO.

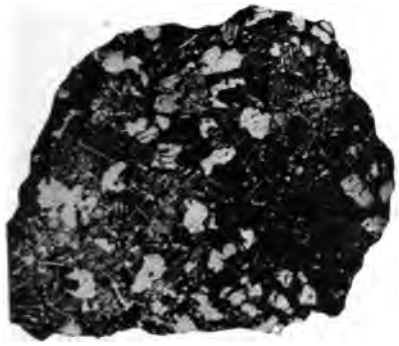
The mineralogical composition of Split Rock ore is presented below:

Mineralogical composition of Split Rock ore and gabbro.

Constituent.	1	2
	Per cent.	*Per cent.*
Ilmenite	2.13	29.42
Magnetite	2.09	22.97
Pyrrhotite	.18	.35
Olivine	16.93	.....
Pyroxene	13.41	22.91
Plagioclase	56.70	13.62
Orthoclase	4.45	.....
Apatite	.34	.....
Calcite	.20	.20
Kaolin	3.10	.....
Water	.18	1.33
NiO.CoO	.02	.....
Spinel	.....	5.97
Corundum	.....	.92
V_2O_3	.....	.55
P_2O_5	.....	.04

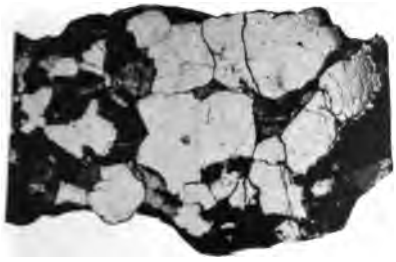
A. SPLIT ROCK, N. Y., ORE. (X 2.)

B. MILLPOND, N. Y., ORE. (X 2.)

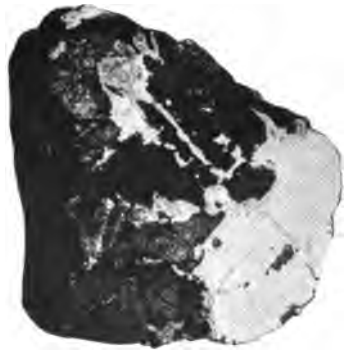


C. TUSCARORA MINE, N. C., ORE. (X 2.)

D. SHANTON RANCH, WYO., ORE (X 2.)



E. CEBOLLA CREEK, COLO., ORE. (X 2.)



F. CEBOLLA CREEK COLO., ORE (X 2.)

D

Jre

The percentage of the iron in the ore is very low and the ore would have to be concentrated before it could be used in the furnace. In that event 5.35 per cent of iron, present in the femic minerals of the gangue, would be lost. It would also be desirable to eliminate as much of the ilmenite as possible, and this carries 10.84 per cent of iron. Only 16.63 per cent of iron is present in the form of magnetite, and so far as possible the concentrates should consist only of it. Notwithstanding the low grade of the ore an unsuccessful attempt was made to work it 30 or more years ago, when a dock was built at the foot of the hill and a small magnetic concentrator erected. The small size of the deposit and the leanness of the ore make the deposit of no value.

METALLOGRAPHIC DESCRIPTION OF ORE.

The ore grains in the lean ore range from 0.1 to 1 mm. in diameter and compose 20 to 50 per cent of the surface. Ilmenite grains are somewhat less abundant than magnetite. The magnetite contains comparatively few inclusions of ilmenite, and these are principally lamellæ, forming an open network structure.

The high-grade ore consists of coarser-grained aggregates of magnetite and ilmenite, ranging up to 3 mm. in diameter, and containing little gangue. The ratio between ilmenite and magnetite is about the same as in the leaner ore. The ilmenite intergrowths in the magnetite are much more abundant. These show in section a closely crowded network of 40 or more lines to a millimeter, as in Plate III, *A*, and a network made up of very short dashes. In much of the ore both types are present in the same magnetite grain and there may be a marked tendency for one series of lines to be more prominent and more fully developed than the others. The effect then is the same as that shown for the Little Pond ore in Plate III, *B*.

LITTLE POND MINES.

Two openings have been made in ore bodies near Little Pond, and 2 miles southeast of Elizabethtown. One opening, about a quarter of a mile north of Little Pond, is 20 by 20 feet and 15 feet deep; the other, lying about an eighth of a mile to the southeast, is run into a hillside about 30 feet and has a width of 30 feet and a working face 25 feet high. The country rock is the normal dark-green gabbro, the exposures of which are numerous in the vicinity. Besides the exposures of ore at the two pits, several others were seen in the neighborhood. The outcrops of gabbro between them show that the ore bodies are a number of small segregations, and not an entire hill of ore inexhaustible in amount as has been stated.

The ore is very similar in character to the Split Rock ore just described. Most of it is a fine-grained ore of seeming good quality, but polished sections show that it contains much gangue. A small

amount of coarser-grained high-grade ore also occurs. The ore also contains the green glass found in the Split Rock mine. The results of two analyses made by W. F. Hillebrand are taken from Kemp, as follows:

Results of analyses of Little Pond ore.

Constituent.	1[a]	2[b]
	Per cent.	*Per cent.*
TiO_2	18.82	13.07
FeO	29.78	28.35
Fe_2O_3	26.30	11.16
Cr_2O_3	.75	.37
V_2O_5	.62	.80
P_2O_5	Trace.	.32
S	.06	.19
	76.33	53.87
Fe	41.57	29.57
Specific gravity	4.410	3.533

[a] Represents ore from the north pit. [b] Represents ore from the south pit.

The analyses have been recast by Kemp, as follows:

Partial mineralogical composition of Little Pond ore.

Constituent.	1	2
	Per cent.	*Per cent.*
Ilmenite	35.27	24.49
Magnetite	38.05	16.24
Chromite	.64	.45
Pyrrhotite	.18	.26

METALLOGRAPHIC DESCRIPTION OF ORE.

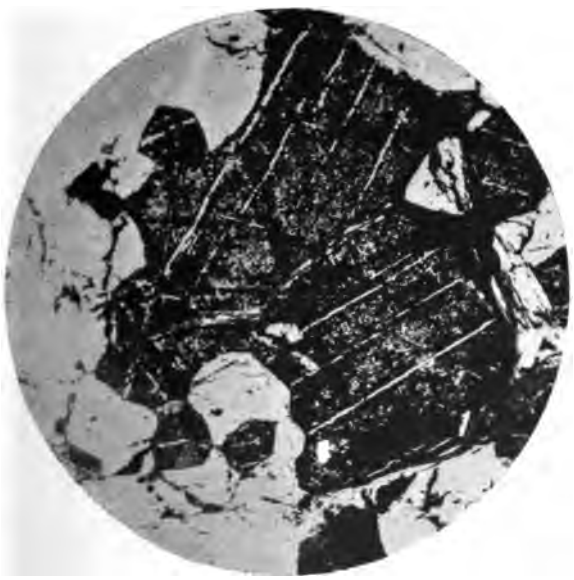
The description given of the Split Rock ore can be applied to this ore with little modification. It averages a little coarser grained and the ilmenite intergrowths in the magnetite are not as closely packed. The more usual spacing is 8 to 15 lines per millimeter. There is also the tendency for one series of lines to be more prominent than the others. An interesting type is where one set of lamellæ are heavy and widely spaced, and the other two sets are made up principally of minute, closely crowded dashes. Such a structure is shown in Plate III, *B*.

TUNNEL MOUNTAIN MINES.

The Tunnel Mountain mines are situated a mile to a mile and a half southeast of Little Pond, and 3½ miles southeast of Elizabethtown. Near the foot of the southeast side of the mountain are two small pits, about 200 yards apart, showing lean ore. Lack of exposures make it impossible to determine the extent of the deposits.

The largest opening occurs on the western edge of the extreme summit of the mountain, where a pit 40 feet long, 10 feet wide, and 40 to 50 feet deep has been made. Two hundred feet below the summit

A. SPLIT ROCK, N. Y., ORE. (× 60.)



B. LITTLE POND, N. Y., ORE. (× 60.)

and south of the pit an adit was commenced and run in about 100 or 150 feet. From this tunnel the mountain received its name. The ore body exposed here has a length of about 300 feet and a width of 100 feet. Beyond the exposure it passes over into the country rock. The latter is described by Kemp as a gneissoid norite with hypersthene the most prominent bisilicate. The other components are green augite, brown hornblende, plagioclase, and garnet. Thin sections of the ore reveal, besides the ore minerals, brown hornblende, serpentinized olivine, garnet, and colorless transparent labradorite.

The ore ranges from medium to fine grained, and like all ore of that kind appears richer in hand specimens than is actually the case. Polished specimens show the presence of a large amount of gangue. The following are the results of two analyses by W. F. Hillebrand, taken from Kemp:

Results of analyses of Tunnel Mountain ores.

Constituent.	1[a]	2[b]
	Per cent.	*Per cent.*
TiO_2	16.45	10.55
FeO	28.52	21.34
Fe_2O_3	20.35	11.52
SiO_2	13.35	.....
Al_2O_3	8.75	.....
Cr_2O_3	.55	.25
CaO	2.15	.....
MgO	6.63	.....
V_2O_5	.61	.34
P_2O_5	.02	.46
S	.09	.10
H_2O	1.68	.....
CO_2	.17	.....
C	Trace.	.....
Cl	Little.	.....
	99.62	.....
Fe	35.99	24.65

[a] Represents ore from the deposit at the top of the mountain.
[b] Represents ore from one of the pits near the foot of the mountain.

These analyses, recast by Kemp, are as follows:

Mineralogical composition of Tunnel Mountain ore.

Constituent.	1	2
	Per cent.	*Per cent.*
Ilmenite	30.80	25.344
Magnetite	29.80	16.704
Chromite	.64	.....
Anorthite	9.45	.....
Spinel	7.38	.....
Olivine	6.94	.....
Enstatite	11.40	.....
Calcite	.40	.....
Water	1.68	.....
Vanadic acid	.61	.....
Sulphur	.09	.....
Phosphorus	.62	.....
Residue SiO_2	.36	.....
Remaining FeO	.....	6.912
	99.57	

METALLOGRAPHIC DESCRIPTION OF ORE.

Gangue constitutes from 25 to more than 50 per cent of the surface of the ore. The magnetite grains are about twice as abundant as the ilmenite, though in places the ilmenite is nearly as abundant as the magnetite. Few grains exceed 1½ mm. in diameter. Regular intergrowths of ilmenite lamellæ in the magnetite are not very abundant, and where they do occur they form a very open structure and are rather thin. Many of the magnetite grains, on the other hand, are swarming with minute inclusions of ilmenite which at times show a tendency to orient along parallel lines, suggesting the incipient development of a network structure. This type of ilmenite inclusion is discussed more fully and illustrated in connection with the description of the metallographic features of the Minnesota ores.

LINCOLN POND MINE.

Three-quarters of a mile southwest of the road crossing the lower end of Lincoln Pond, and a quarter of a mile back from the pond, is an open cut into a steep cliff of gabbro, which is known as the Lincoln Pond or Kent mine. The location is 5 miles northwest of Mineville. The opening is 15 feet wide and nearly 100 feet long. At the far end a shaft was put down, but this is now filled with water, so that its depth could not be determined at the time of inspection. The opening shows little ore, the principal showing being on the left side at the far end. There is a gradual transition from gabbro to ore without any change in the size of the constituent mineral grains. Both ore and rock are fine grained. The rock is described by Kemp as varying from a true norite to a gabbro with accessory hypersthene. Garnet may or may not be present, and in some places is rather abundant. It assumes at times a peculiar fingerlike development in the plagioclase, a feature that has been noticed at other places in the Adirondack gabbros.

About 50 feet beyond this opening is a small opening in which is exposed high-grade coarse-grained ore resembling in appearance the ore in the anorthosites. The contact between this ore and the gabbro is sharp and lacking in a transitional zone.

The following results of analyses of wall rock and ore are available:

Results of analyses of Lincoln Pond ores and wall rock.

Constituent.	1 [a]	2 [b]	3 [c]	4 [d]
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
SiO_2	44.77	11.73	4.73	.....
TiO_2	5.26	12.31	2.58	5.88
Al_2O_3	12.46	6.46	.....	.....
Fe_2O_3	4.63	30.68	.....	.....
FeO	12.99	27.92	.....	.....
NiO.CoO	Trace.	(e)	.....	.....
MnO	.17	(e)	.05	.....
CaO	10.20	3.95	.....	.....
BaO	Trace.	.....	.....	.....
MgO	5.34	3.35	.....	.....
K_2O	.95	.26	.....	.....
Na_2O	2.47	.50	.....	.....
H_2O below 105°	.12	.....	.....	.....
H_2O above 100°	.48	.64	.....	.....
P_2O_5	.28	.82	1.074	.....
V_2O_5	(e)	.04	.....	.....
CO_2	.37	.32	.....	.....
S	.26	.04	.272	.....
C	.....	.05	.....	.....
Cl	.....	.12	.....	.....
F	.....	Trace.	.....	.....
	100.75	99.19	.....	.....
Fe	.....	44.19	59.26	64.66
Specific gravity	3.090	4.138	.....	.....

[a] Represents wall rock. Analyst, George Steiger. (See Kemp, J. F., The titaniferous iron ores of the Adirondacks: 19th Ann. Rept. U. S. Geol. Survey, pt. 3, 1899, p. 407.

[b] Represents ore Analyst, W. F. Hillebrand. (See Kemp, J. F., op. cit, p. 407.)

[c] Represents ore. Figures furnished by W. L. Cumings of the Bethlehem Steel Co.

[d] Represents coarse-grained high-grade ore taken from the small pit. Results were calculated from results of analyses by A. C. Fieldner, of the Bureau of Mines, of concentrates and tailings of a magnetic separation.

[e] Not determined.

The high phosphorus content of the ore is noteworthy and is corroborated by the abundant apatite seen in thin sections. Another feature commented on by Kemp is the presence of carbon. Traces of carbon were found in the ores from some of the other localities, and in this case it amounts to 0.05 per cent. Hillebrand inferred the presence of graphite, though this could not be unquestionably established. Of the three ore analyses, analysis 2 alone represents the normal titaniferous ore in the gabbro. Its iron content is somewhat higher than the average for the region.

Analyses 1 and 2 have been recast by Kemp with the following results:

Mineralogical composition of Lincoln Pond gabbro and ore.

Constituent.	1	2
	Per cent.	*Per cent.*
Ilmenite	9.73	22.95
Magnetite	6.73	44.31
Pyrrhotite	.65	.09
Apatite	.67	1.74
Olivine	2.93	7.38
Pyroxene	32.71	5.01
Plagioclase	37.36	12.53
Orthoclase	5.00	.....
Kaolin	3.60	.....
Calcite	.90	.....
Spinel	.....	3.55

METALLOGRAPHIC DESCRIPTION OF ORE.

The fine-grained lean ore of this occurrence is similar to the ore from other occurrences of the region previously described. On the polished surfaces examined ilmenite intergrowths in the magnetite are not very abundant and consist chiefly of swarms of minute dots. Such line structure as is developed is caused by widely scattered lamellæ, so that the intersecting-line pattern is not prominent. The individual ore grains of the coarser ore reach a diameter of 5 mm. or even more in a few cases, but the average is between 2 and 3 mm. Gangue minerals are almost lacking. Polished sections of this ore differ decidedly in appearance from most coarse-grained titaniferous magnetites. Instead of the sharp contrast between the bright surfaces of the ilmenite grains and the dull faces of the magnetite most of the surface, on being etched, takes on an iron-gray to silver-gray color. Bright surfaces of ilmenite grains occur only sparingly. The duller grains of magnetite show an open-line structure in which the lines are not usually continuous, but are made up of a series of dashes as shown in Plate IV, *A*. This figure also shows the film of ilmenite that frequently outlines the magnetite grains. The more lustrous grains have the silvery, mottled appearance of nontitaniferous magnetites. They differ from such, however, in that in places this mottled effect loses its irregularity and takes on a linear arrangement. Hence the mottling may be due to minute ilmenite inclusions possessing no regular orientation.

The results of a magnetic separation and analysis of the products of a sample of the coarse ore bear out the above metallographic description of it. These results follow:

Results of magnetic concentration and analysis of Lincoln Pond ore.

	Unscreened ore.	Ore through 50-mesh, over 100-mesh.[a]		Magnetic ore through screen finer than 100-mesh.[b]
		Magnetic.	Non-magnetic.	
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Quantity		96	4	95.6
Fe	64.66	66.08	30.59	66.27
TiO_2	5.88	4.82	31.45	5.12

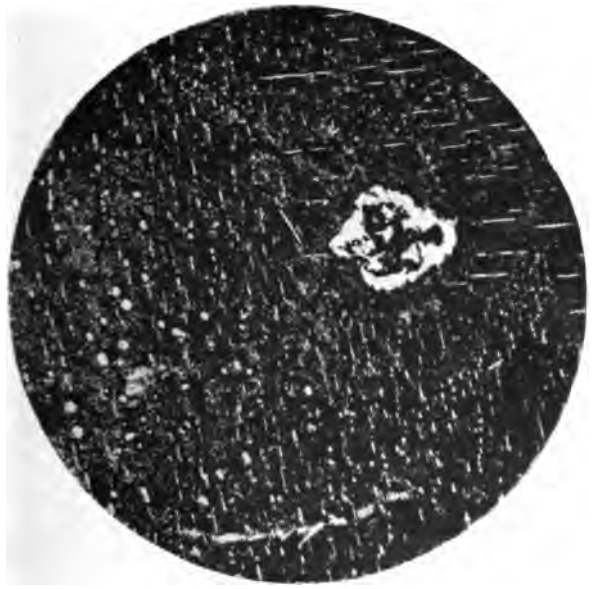
[a] 46.9 per cent. [b] 53.1 per cent.

It is thus seen that the ore is low in titanium and that only 2.4 per cent of ilmenite was separable.

DALTON ORE.

A deposit known as the Dalton ore has recently been prospected by Witherbee, Sherman & Co. by means of diamond drilling and a

A. LINCOLN POND, N. Y., ORE. (× 60.)



B. MILLPOND, N. Y., ORE. (× 60.)

small pit. It is situated about 1 mile northwest of Feeder Pond and 5½ miles west of Mineville, high up on the east slope of Ash Craft Brook. The pit is on the main ore body and is 20 by 10 feet by 5 feet deep. The relations of the ore bodies to the country rock are well exposed here and hence the deposit is of unusual interest. The country rock is a greenish gabbro with considerable dark, nearly black garnet. The garnet occurs in numerous small clusters producing a rock resembling the garnet-rich gabbro of the Split Rock mine. The ore outcrops occur within an area about 300 feet long parallel to the valley and 100 feet wide. At the southwest end of this area the gabbro is cut by a small pegmatite dike.

The ores are much richer than the average of the region and also differ in that there is no transition of gabbro into ore, so that the ore can not be considered an iron-rich phase of the gabbro, but it appears in distinct schlierenlike masses which seem to have been injected into their present position while the gabbro was still in a molten condition. The contact between the ore and the wall rock is marked by a distinct garnet-rich zone. The ores occur in narrow veinlets less than an inch in width, increasing in size to larger masses which, in the case of the main ore body, attain a width of 40 feet. The small veinlets are usually homogeneous in character and free from inclusions of country rock; the larger masses may have irregular outlines and also contain inclusions of country rock. The gabbro in the vicinity is much sheared and the ore tends to lie parallel to the schistosity, though some veins plainly cut across the schistosity.

The ore contains little gangue and this in some places is principally feldspar and at others principally femic minerals. It is about as coarse grained as the ore in the anorthosite, which it much resembles in the hand specimen. The following results of analyses, with the exception of the last, were furnished by Mr. Le Fevre of Witherbee, Sherman & Co. The results last given were furnished by W. L. Cumings of the Bethlehem Steel Co.:

Results of analyses of Dalton ore.

Constituent.	1	2	3	4	5	6	7
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Fe	52.20	47.79	50.46	47.71	48.00	56.03	57.71
TiO_2	7.12	7.53	11.47	5.62	7.58	7.72	3.90
P	.020	.042	.017	.026	.060	.014	.048
S							.022
SiO_2							4.95
Mn							.07

The percentage of iron and titanium in this ore is similar to that of the coarse-grained Lincoln Pond ore. In megascopic appearance they are much alike. Though not as well exposed at Lincoln Pond as

here, it was noticed there also that the contact between rich ore and country rock was sharp and not transitional. Geologically the coarse-grained Lincoln Pond ore would seem to be identical with the Dalton ore.

METALLOGRAPHIC DESCRIPTION OF ORE.

The similarity of the Dalton ore and the coarse-grained Lincoln Pond ore in geologic occurrence, in appearance, and in composition is likewise shown in microstructure. Large grains of ilmenite are more abundant in the Dalton ore, but the appearance of the etched magnetite is nearly the same. Some of the duller magnetite grains show a tendency to rectilinear arrangement of ilmenite intergrowths which are not continuous lines but a series of dashes as illustrated in Plate IV, *A*. The triangular network is faintly developed in a few places. The greater part of the magnetite has the grayish, mottled appearance described in discussing the Lincoln Pond ore.

This ore and the coarse-grained Lincoln Pond ore differ from the other occurrences of the gabbro in this region in the relation of the ore to the wall rock, in the higher iron content and lower titanium content, and in the lack of distinctiveness of the ilmenite intergrowths in the magnetite. Their metallographic features suggest a close relation to the low-titanium ores in the syenite, such as occur at the Nigger Hill mine on the mountain northwest of Lincoln Pond.

POSSIBILITIES OF UTILIZING THE ORES IN THE GABBRO.

All of the deposits in the three townships just discussed occur in the gabbros, and with two exceptions show the same general characteristics. Further, these characteristics differ sufficiently from those of the ores in the anorthosite to make a separate discussion of their commercial possibilities necessary. As the ores in Newcomb Township, which are next described, occur principally in the anorthosite, the economic aspect of the ores in the gabbro is considered at this point.

The facts in regard to the individual deposits have been stated in the foregoing detailed descriptions. It is now desired to give a comprehensive statement of the possibilities of this class of ores considered as a whole. This phase of the question is so well presented by Kemp [a] that his opinion is quoted verbatim in the following paragraphs:

> Enough analyses are now in hand to illustrate in a satisfactory manner what may be expected. The percentage in iron, titanic oxide, phosphorus, and sulphur may be first summarized with the name of the sampler.

[a] Kemp, J. F., Geology of the Elizabethtown and Port Henry quadrangles: N. Y. State Mus. Bull. 128, 1910, pp. 147-149.

Results of analyses of ores from the Adirondack region.

Source of ore.	Sampler.	Fe.	TiO_2.	P.	S.
		Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Split Rock	J. F. Kemp	32.82	15.66	0.017	0.14
Do	G. W. Maynard	32.59	14.70	.....	.....
Tryan Pit	J. F. Kemp	24.65	10.55	.20	.10
Tunnel Mountain	do	35.99	16.42	.009	.09
Little Pond	do	41.57	18.82	Trace.	.06
Do	do	29.87	13.07	.14	.10
Lincoln Pond	do	44.19	12.31	.36	.04
Oak Hill	do	38.98	5.21	.06	.04
Kingdom Works	G. W. Maynard	32.59	13.15	.....	.....
Iron Mountain	J. F. Kemp	40.42	16.37	.....	.....

It is at once apparent that all of these ores are extremely low grade, the richest being 44.19 and only two others reaching 40. Since under present conditions and those which are likely to continue for many years no magnetite under 50 per cent in iron is of importance as a source of lump ore, unless it should have exceptional purity in phosphorus and sulphur, be lacking in titanium, and be in addition located near a furnace, there is little encouragement to look with favor upon bodies of this type.

The percentages in phosphorus and sulphur are also important features. In sulphur the ores are obviously low. In phosphorus they are variable. In instances such as Split Rock and Tunnel Mountain they are very low; in others they are quite high, as at Lincoln Pond. There is a somewhat widely prevalent impression that the titaniferous ores always run low in phosphorus and sulphur, but this is clearly unjustified. As with other ores each case must be sampled by itself.

The presence of vanadium in these ores is a matter of much scientific interest, and since the element has come into such extended use for high-grade steels some have looked to the titaniferous ores as possible sources. If we summarize the results given above, we obtain:

	V_2O_5.
Split Rock	0.55
Tryan Pit	.34
Tunnel Mountain	.61
Lincoln Pond	.62
Little Pond	.50
Do	.04

In just what form the vanadium is combined is unknown. From its chemical properties similar to phosphorus one would suspect some compound analogous to apatite, just as we have pyromorphite and vanadinite, but although the vanadic oxide exceeds in amount the phosphoric, the mineral containing it has never been isolated.

Ferrovanadium is manufactured from vanadium compounds by electrical processes and contains about 25 to 27 per cent of this element. It would appear as if the percentage of this valuable substance were too low to make it a serious factor in the value of the ore, but as the industry of vanadium is as yet in its infancy one should speak regarding its future in a conservative spirit. In a vanadium steel now so higly prized for its toughness there is much less than 1 per cent vanadium. Elementary vanadium constitutes 77.4 per cent of vanadic oxide (V_2O_5).

Magnetic iron ores under 50 per cent and not fulfilling the conditions stated above must undergo magnetic concentration if they are to be utilized. It is with regard to this method of treatment that the recasting of the analyses into percentages of ilmenite and magnetite has especial significance. The magnetite would be the mineral saved and the one upon which efforts would be especially expended. The iron in the ilmenite we would expect, if not hope, to lose, so as to reduce the titanium. The iron

in the pyroxene and olivine would pass off in the nonmagnetic tailings. So far as iron is concerned we are therefore reduced to considering along the magnetite, and therefore the following tabular summary is presented:

Mineralogical composition of Adirondack ores.

Source of ore.	Ilmenite.	Magnetite.	Iron in magnetite.
Split Rock	29.42	22.97	16.63
Do	27.95	32.8	23.74
Tryan Pit	25.34	16.71	12.10
Tunnel Mountain	30.80	29.80	21.58
Little Pond	35.27	38.05	27.55
Do	24.49	16.24	11.76
Lincoln Pond	22.95	44.31	32.08
Oak Hill	9.70	44.08	31.91
Kingdom Works	24.64	31.08	22.50
Iron Mountain, Elizabethtown	30.80	35.73	25.86

These results show that even if the ilmenite and magnetite are so coarsely intergrown as to make a separation feasible the grade of the ore is too low to make the separation a likely source of profit. On the other hand, the ore is extremely hard and fine grained, quite different from the richer and more coarsely crystallized occurrences at Lake Sanford, and parallels can not be justly drawn. While the concentrates would doubtless be somewhat enriched in iron by ilmenite which would enter them, they would be decreased by some inevitable losses in magnetite, and by just so much as the titanium exceeded a very small value, say 1 per cent, the operators of iron furnaces under present slag calculations would regard them unfavorably.

The conclusion is quite irresistible that only by smelting in the crude or lump form, and by the development of a process which does not find titanium objectionable, and under conditions where ores of iron content of 35 to 45 could be utilized, can these deposits be made available.

That these ores hold out small economic possibilities for some time to come can be demonstrated by other considerations not mentioned above. With only one or two exceptions they are at present inaccessible to proper transportation facilities. To provide them with such would involve an expense that would be warranted only in connection with very large deposits. From the nature of the deposits there is no reason to expect the third dimension, or the extension in depth, to be any greater than the larger of the two surface dimensions. Wherever the areal extent of the deposits can be determined it is either small or, if large, its size is not due to the presence of a single large ore body, but to a series of small deposits, between which are areas of barren country rock. Adding to this difficulty the leanness of the ores, which makes the erecting of a concentrating plant necessary, and their titaniferous character, the hope of utilizing these ores within any reasonable period in the future is by no means bright.

DESCRIPTION OF OCCURRENCES IN NEWCOMB TOWNSHIP.

Several titaniferous ore bodies occur in this township, near the headwaters of the Hudson River, in the neighborhood of Henderson and Sanford Lakes. The position of these ore bodies is shown in

Plate V.[a] They include the largest bodies of titaniferous iron ore in the Adirondacks, and one of them, the Sanford ore bed, is exceeded in size by only one other deposit in the United States, that of Iron Mountain, Wyo.

As this region was at one time the seat of the most extensive operations yet conducted on titaniferous ores in this country, a brief historical sketch taken from Newland[b] is of interest here. The sketch follows:

The site of the former Adirondack village (now occupied by the Tahawus Club), which was built by the early ironworkers, lies in the midst of a wild, heavily forested region, shut in by high elevations on all sides except the south, where the river has worn a narrow valley. North Creek, the terminus of the Adirondack branch of the Delaware & Hudson Railroad, is about 30 miles distant by wagon road, and Port Henry, on Lake Champlain, about 50 miles.

Unusual interest attaches to the events connected with the first developments of the Lake Sanford deposits and the establishment of the local iron-making enterprise to utilize the ores.[c] Following the discovery, which is reported to have been made in 1826, a tract of land comprising the deposits was secured from the State by Mr. A. McIntyre and associates who soon after began active work. The investigations of Prof. Emmons in connection with the geological survey of New York then in progress no doubt gave a stimulus to the undertaking. Prof. Emmons published in his reports an extended account of the ore bodies, which he recognized to be of enormous size and regarded as eminently adapted to utilization. He recommended the location of iron-manufacturing enterprises in the vicinity. Soon after the publication of his first report, or about 1840, a blast furnace of 3 or 4 tons daily capacity was built and placed in operation. This was afterwards remodeled so as to enlarge its capacity, and a second furnace of 12 to 15 tons was put in blast in 1854. Drawings of the large stack, which remains to the present day, with all its essential features have been made by Mr. Rossi and published in the article already referred to. The installation included also puddling furnaces and the necessary equipment for making bar iron. The works were closed down in 1856, after which they were not again operated for any length of time. The product of the furnaces was hauled over a difficult mountain road to Crown Point for shipment, and the expense of transportation must have been a heavy tax upon the enterprise.

There seems to be little doubt, judging from all accounts, that the iron turned out in the early days was of good quality; in fact it was especially commended by Emmons and others; nor does it appear that the sudden termination of iron making was due to metallurgical difficulties in reducing the ore, though it is probable that the operators, at least in the early years, were unaware of the titaniferous character of the material. From considerations based on an analysis of slag, which was taken from the dump near the old furnace, Mr. Rossi has expressed the opinion that the furnace charges were made up on somewhat different lines than usually practiced in that a proportion of the country rock (anorthosite) was added to the limestone for flux. It may be noted, however, that the crude ore such as was employed in the operations contains more or less of admixed rock, so that the presence of the latter may have been accidental rather than intentional.

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[a] The map comprising this plate is a reproduction, with slight correction, of Plate 15 of N. Y. State Mus. Bull. 119.

[b] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 3, 1908, pp. 155–156.

[c] A good historical account of the discovery and exploitation of the deposits will be found in Watson's "History of Essex County." The reports by Emmons contain a description of developments up to 1840. For details as to the blast furnaces and metallurgical operations consult Rossi, "Titaniferous Ores in the Blast Furnace": Trans. Am. Inst. Min. Eng., vol. 21, 1892–93.

After lying idle for 50 years the property was taken over in 1907 by a new organization * * * with a view to the exploitation of the ores. This company has conducted a thorough investigation and intends to enter upon active mining in the near future. The construction of a railroad is a requisite before commercial shipments can be made.

GENERAL FIELD RELATIONS.

The area under consideration lies within the main anorthosite area of the Adirondacks, near its western boundary. With one or two exceptions where the ore occurs in gabbros, the anorthosite forms the country rock of the deposits.

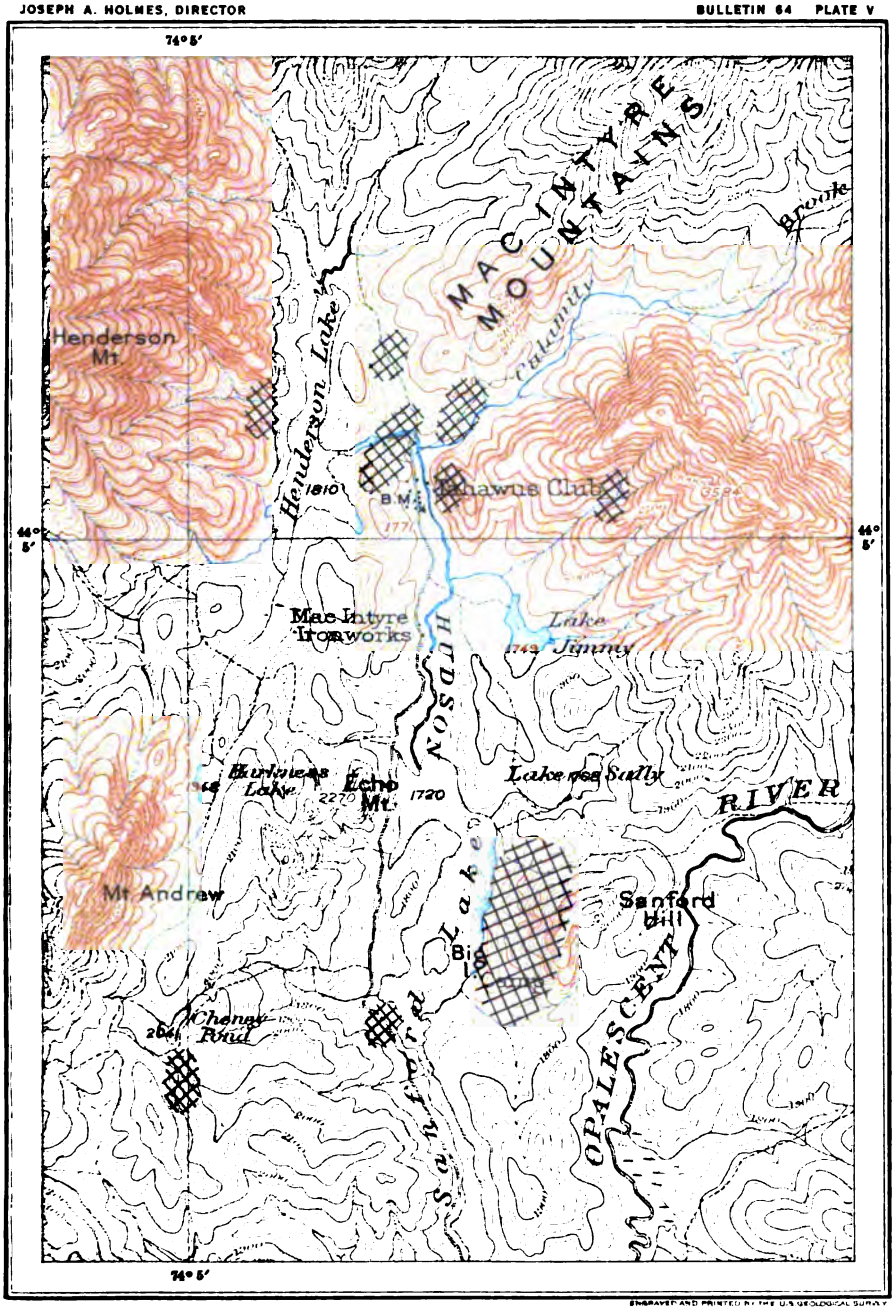
In its typical development, the anorthosite is a nearly pure plagioclase rock in which labradorite is the predominant feldspar. The texture of the rock is coarsely granitoid, and the length of the feldspar crystals may reach several inches. It is most commonly light blue in color, but at times it is yellow and even black. Usually little else than the feldspar can be recognized macroscopically. In thin section, however, augite, hypersthene, and magnetite are seen as accessory constituents in widely varying amounts. Magnetite and ilmenite are far less abundant in this rock than in the gabbros. The anorthosite has resisted crushing and mashing to a far greater extent than the gabbros, and gneissic varieties are not so common. Where crushing has taken place, garnet, biotite, and calcite are developed as secondary minerals. No results of analysis of the anorthosite of this area are available, but the analysis whose results are given below was made by Prof. Albert R. Leeds[a] of a sample taken from the summit of Mount Marcy.

Results of analysis of anorthosite from Mount Marcy.

Constituent.	Per cent.
SiO_2	54.47
Al_2O_3	26.45
Fe_2O_3	1.30
FeO	.67
CaO	10.80
MgO	.69
Na_2O	4.37
K_2O	.92
Loss	.53

In a few places within the ore-bearing area the typical anorthosite gives way to a fine-grained rock in which the femic minerals are more abundant, forming a gradation to gabbro. This rock seems to have been less resistant to metamorphism and in many places shows a gneissic structure. In such cases it contains considerable hornblende derived from the pyroxene and much garnet. The rock is undoubtedly a phase of the anorthosite, but in some cases is not a differentiation in situ but has been intruded into its present position in the anorthosite in the form of dikes.

[a] 30th Ann. Rept. N. Y. State Mus., 1878, p. 92.



ENGRAVED AND PRINTED BY THE U.S. GEOLOGICAL SURVEY

MAP OF THE VICINITY OF LAKE SANFORD, NEW YORK

ORE BODIES ARE INDICATED BY CROSS HATCHING

Scale $\frac{1}{62500}$

1 ½ 0 1 2 Miles

DESCRIPTION OF ORE DEPOSITS.

Ore occurs in both the anorthosite and the gabbro, but in this area the anorthosite is the more common wall rock. The ores in the gabbro are very similar to the ores in that rock in Westport, Elizabethtown, and Moriah Townships. Their geological occurrence and their chemical composition are about the same.

The ores in the anorthosite are of more uniform character and of higher grade in iron, and also coarser grained. The gradual transition from wall rock to ore so common in the gabbros is usually lacking. Anorthosite and ore are in most cases in sharp contact, with little if any increase in the amount of femic minerals in the anorthosite as the ore is approached. There seems to have been a complete differentiation of the ferruginous and titaniferous minerals on the one hand and the feldspathic on the other. In the one rock, iron oxide and titanium oxide crystallized by themselves; in the other, silica, alumina, soda, and lime. The intermediate femic silicates formed in small amount, a condition that Kemp suggests was due to the lack of magnesia. The ore carries a very small percentage of gangue, but in many places contains crystals of labradorite, which may attain considerable size. The feldspar is, however, never in actual contact with the ore minerals, but is separated from them by a narrow zone of fine-grained femic silicates. Further details in regard to the character of the deposits are given in the descriptions of the individual deposits.

DESCRIPTION OF DEPOSITS IN GABBRO.

CHENEY POND ORE.

The Cheney Pond deposit is situated near the south end of Cheney Pond, a little over a mile west of the road running along Sanford Lake. About 350 yards south of the lake is a pit 35 feet wide and 20 to 30 feet deep. Between this pit and the lake several diamond-drill holes were put in a few years ago. The cores of the drill supply information not otherwise obtainable on account of the lack of fresh rock exposures.

The rock is a gabbro gneiss which is very crumbly on weathered surfaces. Garnet and magnetite are prominent constituents throughout the rock. With increasing amounts of magnetite the rock grades over into ore. A thin section of the average grade of ore shows the ore minerals and plagioclase as the most abundant constituents, with the ore minerals forming about 50 per cent of the rock. Augite and garnet are also prominent. Less abundant are biotite, apatite, and dark-green spinel. The garnet and pyroxene tend to occur around the periphery of the feldspar crystals. Frequently the augite is replaced to a great extent by hornblende. Though most

of the ore is rather lean, there is a little high-grade ore in which there is only a small amount of gangue. The drill cores show that the gabbro grades over into anorthositic rock, so that at this locality the gabbro is merely a basic phase of the anorthosite and not an intrusive dike.

The results of two analyses of the ore from the pit have been published. One of these represents the average lean ore; the other must have been made from a sample of the richer phase and does not represent the run of these deposits.

Results of analyses of Cheney pit ore.

Constituent.	1	2
	Per cent.	*Per cent.*
TiO_2	8.25	15.77
Fe_3O_4	86.53	55.64
SiO_2	.....	9.79
Al_2O_3	.....	7.12
CaO	.....	8.89
MgO	.....	3.00
P	.39	.....
S	.74	1.00
Fe	62.15	49.33

The second analysis represents about the same composition as the ores in the gabbro in the townships to the east described on preceding pages.

Metallographic description of ore.—The microscopic structure of the ore presents little of interest. The size of the grain is roughly proportional to the richness of the ore. In the higher grade ores, ilmenite and magnetite grains attain dimensions of 4 mm.; in the leaner ores they scarcely reach 2 mm. and average considerably less. The ratio of ilmenite to magnetite is variable; some sections show nearly all ilmenite, others only a small amount. The average of all the sections examined shows the ilmenite somewhat in excess of the magnetite.

A surprising feature of the magnetite is the almost entire absence of ilmenite lamellæ intergrown in it. Only here and there was a line or two seen in the magnetite. On the other hand the swarms of minute ilmenite particles possessing no regular orientation and giving the magnetite a gray, mottled appearance are abundant. Besides these there occur large circular dots in dull magnetite grains.

On the whole what has been said regarding the possibilities of utilizing the ores in the gabbro applies to this deposit, except that it seems to be considerably larger than the average.

OTHER OCCURRENCES IN GABBRO.

Exposures of iron ore in fine-grained gabbro occur along Calamity Brook for some distance above its mouth. The ore is first encountered about 200 feet above the mouth of the stream on the north side,

where it plainly cuts the anorthosite. About 100 feet above this point another dike cuts across the brook in a northeasterly direction. For two or three hundred yards there is an alternation of this fine-grained rock and anorthosite. In places the contact between the two is a straight line, in others the boundary is irregular. In either case, however, the boundary is usually sharp and lacking in a transitional zone. At some of these contacts the evidence that they are intrusive is unmistakable. The rock is characterized by an abundance of magnetite and ilmenite and by the small amount of feldspar that it contains. Considerable pyrite is also present. A thin section of a richer phase of the rock shows besides the ore minerals augite and hypersthene as the most prominent constituents, a little biotite and spinel being noticeable. Locally the ore minerals become abundant enough to consider the rock a lean ore. On account of the fine grain and the lack of salic minerals, the rock has a black appearance which makes it look to be much richer than it really is.

The occurrence one mile east of the Tahawus Club, high up on the mountain slope, is more like the Cheney Pond ore in that it carries considerable plagioclase.

Some of the ore on the north side of the Hudson River between the bend and Lake Henderson is fine grained and lean, due to the presence of abundant femic minerals. This is especially so toward the west end of the ore-bearing area. Some of the drill cores show an alternation of fine-grained lean ore and coarse-grained high-grade ore. The following section of one of these cores is given by Newland:[a]

Section from diamond-drill core from bore hole on north side of the Hudson River near Lake Henderson.

	Ft.	in.
Lean ore, consisting of disseminated magnetite with feldspar and pyroxene..	19	3
Rich ore	30	6
Rock and lean ore, alternating	15	..
Rich ore	33	6
Rock carrying some ore	11	1
Lean ore	12	..
Rock and lean ore	8	..
Rich ore alternating with seams of rock	10	7
Rock	5	..
	144	11

Though not so stated, what is called lean ore in this record is doubtless of the fine-grained gabbroitic variety.

DESCRIPTION OF ORES IN ANORTHOSITE.

The ores in the anorthosite attain their chief development along the Hudson River between the Tahawus Club and Henderson Lake

[a] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 3, 1908, p. 162.

and on the west side of Sanford Hill overlooking Sanford Lake. The former area was the principal source of ore for the early iron works operated here, and the pit from which the ore was taken is called the Millpond pit.

MILLPOND ORE.

The Millpond pit is on the west side of the river directly opposite the mouth of Calamity Brook. It is an opening about 100 feet long and 12 to 40 feet wide, now filled with water. The lower part of the walls of the opening just above the level of the water shows a coarse-grained high-grade ore, nearly free from visible gangue minerals. The upper part of the walls around the rim of the opening consists of typical anorthosite. The contact between the two is, therefore, well exposed. The boundary is irregular in its outline, but sharp. Pure ore usually extends up to the very contact. The anorthosite at times contains small patches of ore surrounded by reaction rims. The intermingling, however, is by no means sufficient to suggest a transition from ore to wall rock.

The reaction rims between the feldspar and the magnetite are most beautifully developed both around feldspar inclusions in the ore and around ore inclusions in the feldspar in this area and on Sanford Hill. The nature of the rims varies considerably. A thin section from the Millpond pit showed a well-developed three-zone reaction rim. The large feldspar crystals swarmed with minute green spinels and acicular pyroxenes to such an extent as to render the feldspar so cloudy that the twinning bands are nearly obscured. Around this nucleus is a zone consisting of clear feldspar, pyroxene, and larger spinel individuals. The feldspar and pyroxenes occur as a matrix inclosing the abundant spinel. The second zone consists of a clear monoclinic pyroxene with very faint greenish tinge. Along fracture and cleavage lines a small amount of magnetite occurs. The third zone consists of a pyroxene of a darker green color, the fractures of which are heavily filled with magnetite. Beyond this zone is the magnetite. The width of the rim where the three zones are best developed is 1 mm.; where it becomes wider the zones lose their distinctness.

Another thin section from a specimen across the river from the Millpond pit showed a different development, though it, too, had three distinct zones. The feldspar is likewise clouded by minute inclusions, but running through it in long fingerlike and vermicular shapes are garnet crystals. The feldspar immediately adjacent to the garnet and, where the garnet had developed abundantly, whole areas of it are perfectly clear, indicating that the garnet is the product of a reaction between the feldspar and the minute inclusions. Surrounding these nuclei of feldspar is always a zone of clear feldspar and garnet. Next comes a narrow zone of clear feldspar nearly

free from any other minerals. In the third zone feldspar is nearly lacking, and the principal minerals are augite, biotite, and chlorite. Beyond this zone is the magnetite, which contains abundant inclusions of dark-green spinel. As in the previous case, the zones are most distinct where the reaction rim is not too wide.

Surface exposures of the ore do not indicate as extensive an area as shown by Newland (Pl. V). Fifty feet west of the Millpond pit is an exposure of anorthosite that is practically continuous to the top of the small knoll between the river and Henderson Lake. It may be that Newland's boundary is based on magnetic surveys. At the Millpond pit the ore is overlain by anorthosite, and it is quite possible that the ore body continues westward under the knoll of anorthosite.

The "Iron Dam" is a riffle across the river a short distance below the Millpond pit, on the surface of which considerable ore is exposed. The ore is of the same coarse-grained character as that of the Millpond pit, but large feldspar phenocrysts are more abundant. It occurs in the form of several larger lenticular bodies and numerous small stringers running through the anorthosite. The relations at the contact of ore and country rock are the same as those described as existing at the Millpond pit. On the east side of the river, between Calamity Brook and the sharp bend above it, are numerous exposures; most of them are of anorthosite. At several places, however, the anorthosite contains small lenticular and irregular shaped ore bodies and stringers of high-grade coarse-grained ore, exhibiting the same relations as the similar occurrences at the "Iron Dam."

The contact between the ore and the wall rock is exposed so often and so well at the localities just described that one can not help trying to picture to himself just how the ore came into its present position. In the processes of magmatic differentiation, as one ordinarily thinks of them, there is a gradual localization of certain constituents of the magma until a product is produced characterized by the predominance of those constituents. With increasing distance from such a differentiated mass, those constituents should more and more decrease and the remaining constituents increase in quantity until the average composition of the magma is again obtained. This change may take place rapidly, or it may take place slowly. One has the feeling, however, that such a transition should exist. In looking at these exposures no such picture is suggested. If a differentiation has taken place, it has been complete. One can put his finger on almost any point along the contact and say this is ore or this is anorthosite. It is difficult to get away from the suggestion that the ore has been injected into the anorthosite. Yet the irregular form of some of the ore bodies, and the numerous stringers running through the anorthosite in all directions present difficulties to explain. To have afforded passages for such irregular injections, the anorthosite must have undergone a certain amount of crushing and shattering.

One of the commonest forms of inclusions shows on the polished section as dots which are scattered over the surface in varying amounts. In places they occur rather sparsely; elsewhere the surface may be thickly covered with them. Where most abundant, they are usually of minute size. They differ from the swarms of minute particles mentioned in the preceding paragraph in that they have distinct, somewhat circular outlines, whereas the latter have indistinct, hazy outlines. Some of the dots attain considerably larger size and reach a diameter of 0.05 mm. The minute dots in such unusual abundance as to produce a clouded effect are seen in Plate IV, *B*. Interspersed among these are also some of the larger sized dots.

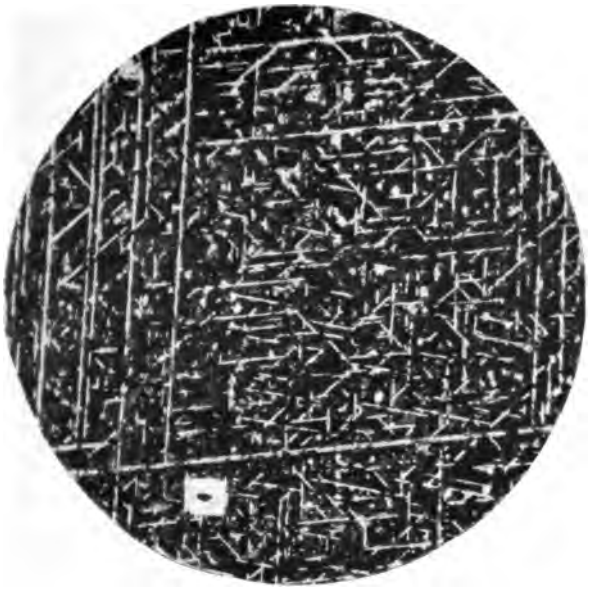
Less abundant, yet prominent, are the tabular intergrowths. More commonly the individual lamellæ are rather short, and where a number occur in the same plane the polished section shows a broken line. The lamellæ may occur isolated, and in such cases often become thicker in proportion to their length than the average. When abundant they take on the regular parallel rectilinear arrangement, producing a one-line, two-line, or three-line intersecting pattern according to the orientation of the section. This broken-line pattern is shown in Plate IV, *B*. The lamellæ are not, however, always short, but attain a length of 1 to 2 mm. and even more in places, producing a continuous-line pattern. Plate VI, *A*, shows a number of these larger lamellæ. The section is also of considerable interest in that it shows the magnetite to be twinned. If examined closely, it will be seen that instead of three series of intersecting lines corresponding to the octahedral cleavage of magnetite there are five series. They correspond to the cleavage of a twin about the octahedral face, and are either four or five in number according to the orientation of the section.

Another feature of many of the sections is the occurrence of a narrow rim of ilmenite about the outlines of the magnetite grains. This phenomenon was already noted in the case of the Lincoln Pond ore (Pl. IV, *A*).

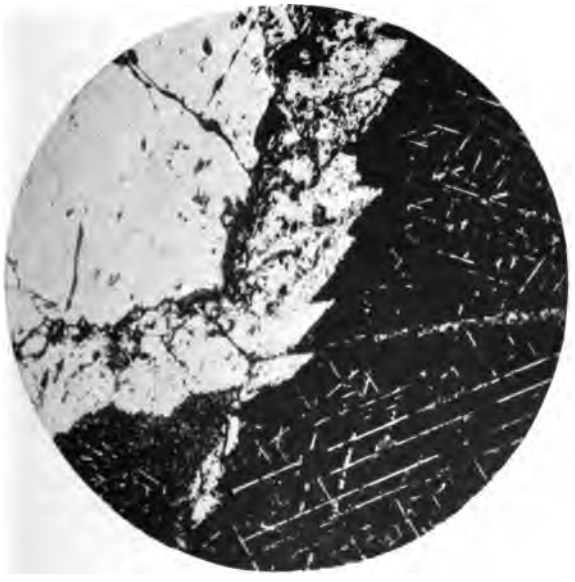
In many places a much coarser form of inclusion than those just described occurs around the ilmenite grains. It is in the form of a zone made up of irregular particles of ilmenite with their longer direction at right angles to the ilmenite border. Locally these particles coalesce, forming a continuous band. There is always a narrow strip of magnetite between the ilmenite grain and this zone. In a few cases the zone completely encircles the ilmenite grain, but usually it is discontinuous. This effect is shown in Plate VI, *B*.

SANFORD HILL ORE.

The Sanford Hill ore is the largest and most important body of titaniferous ore in the Adirondacks. It occurs on the west slope of



A. IRON DAM, N. Y., ORE. (× 60.)



B. IRON DAM, N. Y., ORE. (× 60.)

Sanford Hill overlooking Sanford Lake. Lack of exposures makes impossible the determination of the limits of the ore body. Diamond drilling during the years 1906 and 1907 has demonstrated that the ore continues practically to the lake. Magnetic surveys show lines of attraction that cross the lake and merge with the small ore body on the western shore below Big Island (Pl. V). It may be that the ore continues under the lake to the western shore. On the east, anorthosite is encountered before the summit of the hill is reached, and the summit itself is composed of anorthosite. The lateral extent, parallel to the lake, has not been determined, but is stated by Newland to be at least a half mile. Drilling has proved its continuance in depth to 300 or 400 feet, the maximum depth reached by the drill.

This part of the hill was laid bare at the time the drilling was done preparatory to commencing stripping, and a few trenches were made. Besides these there is an old opening on the hillside with a working face about 30 feet long and 15 feet high.

The anorthosite exposed on the east side of the deposit is very gneissoid. The feldspar is nearly white and running through it are black bands which in thin section are seen to be composed of dark-green hornblende. Smaller particles of the hornblende are scattered through the feldspar. Anorthosite is also scattered within the boundaries of the ore body. It seems to be segregated along certain bands and could easily be sorted in mining so as not to impair the quality of the ore. A thin section of the rock showed a small amount of augite that has to a great extent altered to hornblende.

In the early forties the deposit was prospected with considerable care by Prof. Emmons, for whom a number of pits were made to expose the ore. These pits have since filled up and their exact locations are not known, but the details of the observations are still valuable in that they show the manner in which the ore and the country rock are intercalated. The most detailed of the sections across the ore body made by Prof. Emmons is here reproduced as given by Newland.[a]

Record of the middle transverse section of the Sanford Hill ore body.

Pit No.	Interval.	Remarks.
	Feet.	
1	.....	Fine granular feldspar, intermixed with iron, garnet, and hornblende.
2	36	Rich ore breaking into tabular masses.
3	10	Do.
4	15	Rich ore.
5	20	Rich ore, mixed in a small proportion with granular feldspar.
6	12	Granular feldspar in a decomposing state containing only a small proportion of ore.
7	20	Rich ore, mixed with a few scales of black mica and feldspar.
8	22	Rich ore, mixed with garnet and feldspar.
9	24	Nearly the same as No. 8, but brighter.
10	24	Rich ore with a very small proportion of feldspar.

[a] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 3, 1908, pp. 159–160.

Record of the middle transverse section of the Sanford Hill ore body—Continued.

Pit No.	Interval.	Remarks.
	Feet.	
11	22	Loose decomposed rock.
12	17	Rich ore.
13	15	Rich ore, with feldspar.
14	39	Rich granular ore, with a resinous luster.
15	15	Lean ore.
16	22	Principally rock.
17	28	Pure ore.
18	35	Do.
19	36	Rich ore.
20	22	Pure ore.
21	27	Do.
22	30	Do.
23	29	Do.
24	30	Ore mixed with garnet.
25	14	Rock mixed with particles of ore.

On the basis of his explorations, Prof. Emmons estimated the ore body to contain 6,830,000 tons, at a depth of 2 feet below the adjoining surface. Besides being a very large deposit, it is well situated for mining, as its topographic position on the side of the hill makes open-quarry methods applicable.

The ore is identical in character with that at the Millpond pit. At the old opening it contains many large phenocrysts of green feldspar, which attain a length of 6 cm. and more. These are always surrounded by a reaction rim in which a golden-brown biotite is a prominent constituent. In thin section the feldspar is clouded by the minute inclusions described in the feldspar at the Millpond pit. Surrounding it is a narrow zone of clear feldspar with larger green inclusions, many of which are spinels. Then comes a broader zone of femic minerals, of which the most abundant are brown mica and hornblende. Less abundant is a nearly colorless augite. Beyond this zone is the magnetite.

In the following table of analyses, analyses 1 to 5 are quoted from Kemp and 6 and 7 from Newland, by whom the original sources are acknowledged. Analysis 8 was calculated from analyses of concentrates and tailings made by A. C. Fieldner of the Bureau of Mines.

Results of analyses of Sanford Hill ore.

Constituent.	1	2	3	4	5	6	7	8
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
TiO_2	10.91	20.03	19.52	18.70	14.52	9.45	14.0	19.02
Fe_3O_4	87.60	70.73	70.80	71.03	.....	.....	83.4	.....
SiO_2	.87	2.46	1.39	1.34	1.39	2.09	.....	.....
Al_2O_3	.53	3.50	4.00	.....	5.81	.....	.....	.....
P	.....	.....	.022	.....	.....	.007	.....	.....
S	.....	.....	.028	.....	.....	.027	.....	.....
Fe	62.65	51.22	51.30	51.44	56.60	63.36	60.5	55.07

In these analyses the average percentage of Fe is 56.52 per cent and of TiO_2 15.77 per cent. Computing the TiO_2 in terms of ilmenite and the remaining iron as magnetite we get 29.96 per cent ilmenite and 62.81 per cent magnetite in the ore, a total of 92.77 per cent. The composition is almost identical with that of the average of the Millpond ore. The two determinations for phosphorus show the content of that element to be below the Bessemer limit; the sulphur content is also low.

Metallographic description of ore.—The microstructure of the Sanford Hill ore is identical with that of the Millpond ore. The same types of intergrowth are found in this ore, and it is therefore useless to repeat the descriptions of them at this point. The only difference, if any, is that the intergrowths of ilmenite in the magnetite are somewhat less abundant in the Sanford Hill ore.

POSSIBILITIES OF UTILIZING THE ORES IN ANORTHOSITE.

The far greater size of the deposits, the much higher iron content, and the coarser texture of the ores in anorthosite make them far more attractive from a commercial standpoint than the ores found in the gabbro. Transportation facilities would have long since been provided if there had been any feeling of certainty that the difficulties caused by the presence of titanium could be overcome. Numerous experiments have been made to separate the titanium from the ore by means of magnetic separation, and the results of some of them are given below.

Kemp [a] gives the following results of analysis of concentrates and tailings of a magnetic separation of the Sanford Hill ore:

Results of magnetic separation of Sanford ore.

Constituent.	Concentrates.	Tailings.
	Per cent.	*Per cent.*
TiO_2	4.00	47.50
Fe	62.66	36.86

No statement is made as to the size to which the ore was crushed nor of the ratio between concentrates and tailings. If we assume the average composition of the Sanford Hill ore as the composition of the sample tested, the ratio of concentrates and tailings can be calculated. On the iron basis the tailings constitute 25.9 per cent of the original ore, and on the titanium basis 27.7 per cent. The agreement is so close that the mean 26.8 per cent may be assumed to be the amount of the tailings.

[a] Kemp, J. F., The titaniferous iron ores of the Adirondacks: 19th Ann. Rept. U. S. Geol. Survey, pt. 3, 1898, p. 416.

The possibilities of magnetic separation of the Sanford Hill ore are also discussed by Newland,[a] whose statements are next given, as follows:

A partial separation of the magnetite and ilmenite was obtained with the ore from the Sanford pit at Lake Sanford. A sample was crushed through a 40-mesh sieve and the magnetite removed with a small hand magnet. The results from chemical analysis of the crude ore (1), magnetite concentrate (2), and the ilmenite and other residual minerals (3) are given herewith. The analyses were made by E. W. Morley.

Results of analyses of Sanford pit ore.

Constituent.	1	2	3
	Per cent.	*Per cent.*	*Per cent.*
Fe_2O_3	55.9	54.39	14.28
FeO	27.5	28.66	30.93
TiO_2	14	8.93	45.23

It will be observed that the magnetite concentrate still contains a considerable proportion of titanium, mostly due, no doubt, to the inclusion of particles of mixed character. By crushing still finer a cleaner separation may be made, as has been demonstrated * * *. The analyses are not reducible to simple terms of magnetite and ilmenite, and further work is needed before the chemical relations can be fully stated. It is quite likely that the magnetite itself carries a proportion of the titanium, in which case the entire removal of the latter would be impossible.

Another feature which may promote the use of the ores from that locality is their amenability to concentration, whereby the titanium can be reduced by at least a third of the total or to less than 6 per cent probably as a maximum limit. To the courtesy of F. E. Bachman, general manager of the Northern Iron Co., the writer is indebted for information concerning an experimental run made upon Lake Sanford ores at the Mineville magnetic concentrating plant within the past year. A 40-ton sample was passed through one of the mills and the concentrates showed the following percentages: Fe, 60.60 per cent; TiO_2, 9.66 per cent (equivalent to 5.8 per cent Ti). The tailings from the treatment gave: Fe, 42.84 per cent; TiO_2, 32.22 per cent. A sample of the concentrates recrushed so as to pass through a 16-mesh screen and reconcentrated by hand showed the following percentages on analysis:

Fe	63
SiO_2	1.08
TiO_2	5.25
Al_2O_3	5.65

Another sample was crushed through a 40-mesh screen and subjected to separation under water with a hand magnet. Analyses of the concentrates are given below. No. 1 is by A. S. McCreath & Son and No. 2 by P. W. Shimer.

Results of two analyses of Lake Sanford ore.

Constituent.	1	2
	Per cent.	*Per cent.*
Fe	65.35	64.69
Mn	.186	.....
SiO_2	.14	.....
TiO_2	5.32	6.49
Al_2O_3	2.76	2.59
CaO	.05	.....
MgO	.76	.....
P	.012	.0045
S	.041	.....

[a] Newland, D. H., Geology of the Adirondack magnetic iron ores: N. Y. State Mus. Bull. 119, pt. 2, 1908, pp. 152, 154–155.

While it would scarcely be practicable, perhaps, to crush the ore to a size that would permit a reduction of the titanium to the limits indicated in the last analyses, there would appear to be no difficulty in the way of preparing concentrates with an average of 8 or 10 per cent TiO_2. The loss of iron in the tailings is the only drawback to concentration, but in the case of immense deposits like those at Lake Sanford, which can be worked very cheaply, this could hardly be critical.

In none of the above cases is the ratio of tailings to concentrates given. In the case of the 40-ton sample, however, a ratio can be calculated by again assuming the original ore to have had the average composition of the Sanford ore, an assumption that, in view of the large quantity used, is fair. On the iron basis the tailings constituted 23 per cent of the original ore and on the titanium basis 27 per cent. The mean of these, 25 per cent, should be nearly correct.

Two separations made by the author, one on Sanford Hill ore and one on Millpond ore, did not give results as favorable as those above. The analyses of the tailings and concentrates were made by A. C. Fieldner, of the Bureau of Mines. The results follow:

Results of magnetic separation of Sanford Hill ore.

	Unscreened ore.	Ore through 50-mesh, over 100-mesh.[a]		Ore through screen finer than 100-mesh.[b]
		Magnetic.	Non-magnetic.	Magnetic.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Quantity	.....	78.2	21.8	75.9
Fe	55.07	61.23	32.99	61.23
TiO_2	19.02	11.15	47.20	10.47

[a] 47 per cent. [b] 53 per cent.

Results of magnetic separation of Millpond ore.

	Unscreened ore.	Ore through 50-mesh, over 100-mesh.		Ore through screen finer than 100-mesh.
		Magnetic.	Non-magnetic.	Magnetic.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Quantity	.....	83.2	16.8	85.3
Fe	54.47	58.89	32.74	59.17
TiO_2	18.20	13.18	43.03	12.28

A sample was also prepared from the Sanford Hill ore by picking small pieces of magnetite from the surface of etched polished sections. As far as possible only such pieces were taken as were free from all visible intergrowths of ilmenite. The sample was further tested with

a magnet to make sure that it contained no nonmagnetic particles. The analysis of this sample made by A. C. Fieldner is as follows:

Fe	61.04
TiO_2	11.00

From the manner in which this sample was prepared, the analysis results given ought to represent the most favorable results obtainable.

In the first two separations made by the writer the percentage of titanium was reduced by 41.4 and 27.6 per cent, respectively. If we assume in the third case the average composition of the Sanford Hill ore, the reduction was 30.3 per cent. The average of the three is 31.1 per cent, which accords with Newland's estimate that the titanium content can be reduced by a third.

The result obtained in the test at the Mineville magnetic-concentrating plant represents, therefore, the maximum separation obtainable; and the average of a long run of the ores is not likely to fall far behind this. A conservative statement of the results obtainable by magnetic separation is that concentrates can be produced running about 60 per cent Fe, less than 7 per cent Ti, low in sulphur, and far below the Bessemer limit in phosphorus, with a loss of not more than 25 per cent in the tailings. Aside from the titanium content, this is a most desirable product for the blast furnace. By mixing these concentrates with other ores in a ratio between 1 to 2 and 1 to 3, the titanium content can be brought down to 2 per cent, a quantity that would cause no trouble in the furnace. The size and situation of the deposit with reference to easy mining methods are such that, with transportation facilities provided, the concentrates could profitably be put on the market at the current prices. Of course the prejudice against titaniferous ores would retard and probably make impossible the sale of the ores if put on the market by an independent operator. There is no reason, however, why the deposits should not be worked by interests that control their own furnaces, and it is doubtless a question of only a few years before mining will have been begun on these ores.

NEW JERSEY.

Magnetite ores occur throughout the pre-Cambrian rocks of the New Jersey Highlands. Many of these ores contain a small amount of titanium, but in only a few instances does the percentage of titanium become as high as that usually contained in titaniferous magnetites. Furthermore, the occurrences with high titanium content are among the smaller deposits of the region, and will never be a factor in the utilization of such ores. The region was not visited for that reason; but because frequent mention is made in the literature of the titanium content of the New Jersey ores a brief account of the deposits will be given.

Practically all of the information available in regard to these ores has been included in the report on "Iron Mines and Mining in New Jersey," by W. S. Bayley,[a] in which reference is made to the original sources.

FIELD RELATIONS OF REGION.

The pre-Cambrian rocks of New Jersey consist of gneisses, pegmatites, and a series of sedimentary rocks called the Franklin formation, of which the most important member is a white crystalline limestone.

These pre-Cambrian rocks form a series of ridges trending northeast, separated from one another by intervening valleys of Paleozoic rocks, and having a general southeast dip. Corresponding to the principal belts or ridges, the magnetites have been grouped into four belts, known from southeast to northwest as the Ramapo, the Passaic, the Musconetcong, and the Pequest. Magnetic iron ore occurs in all of the pre-Cambrian rocks, but the titaniferous ores are confined to the gneisses. The gneisses consist of alternations of potash rich, soda rich, and iron and magnesium rich rocks, called, respectively, the Byram gneiss, the Losee gneiss, and the Pochuck gneiss.

The origin of the gneisses is rather problematical, as they have been subjected to intense regional metamorphism. Wolff[b] made a microscopic study of the gneisses at Hibernia and favors an origin by metamorphism of sediments. Westgate,[c] who studied the rocks on Jenny Jump Mountain, is in doubt about the light-colored feldspathic gneisses, but favors an igneous origin for the dark hornblendic varieties. Bayley[d] compares an analysis of a specimen of the hornblendic variety with an analysis of an Adirondack norite, and finds them chemically almost identical. Hence, though there is evidence pointing to the existence of metamorphosed gabbroitic rocks, the relation of the titaniferous ores to such rocks is not yet made clear. The titaniferous ores all occur within the areas mapped as acid gneisses, and not within the areas of the Pochuck gneiss. It seems, however, that intercalations of Pochuck gneiss too small to map occur in the acid gneisses, as the presence of such is mentioned at the Richard mine, though none is indicated on the map at that point. In other cases the character of the gneiss is not made clear.

Direct evidence that any of these ores are magmatic segregations in gabbros that have been subsequently metamorphosed is lacking. The very low titanium content of nearly all of the ores, with every gradation down to ores practically free from titanium, compared to the uniformly high titanium content of magmatic segregations of

[a] Vol. 7 of the final-report series of the State geologist, 1910.

[b] Wolff, J. E., Geological structure in vicinity of Hibernia, N. J., and its relation to the ore deposits: Ann. Rept. of State Geologist of New Jersey, 1893, pp. 359–369.

[c] Westgate, L. G., The geology of the northern part of Jenny Jump Mountain, Warren County: Ann. Rept. of State Geologist of New Jersey, 1895, pp. 21–61.

[d] Op. cit., p. 123.

titaniferous magnetites must be of some significance. It has long been believed, and the belief is founded on experience, that only magnetites segregated in gabbroitic rocks are titaniferous, or carry more than a very small amount of titanium; and when analyses of some of these New Jersey ores were found to run high in titanium it was only natural to try to connect those ores with rocks of that character. A notable exception to such a rule is the contact-metamorphic titaniferous magnetite on the Cebolla Creek in Gunnison County, Colo.[a] It is true that the source of the ore on the Cebolla is a basic rock, nevertheless the method of origin was pneumatolytic, showing that considerable titanium can be carried away from the magma by pneumatolytic processes. A review of the theory of origin advanced by Bayley to account for the New Jersey magnetites is beyond the scope of this report, and for the present purpose it is characterized as of combined magmatic and pneumatolytic, or hydato-pneumatolytic, origin. The gneisses are regarded as of igneous origin. After their intrusion and partial cooling ferruginous portions of the same magma were intruded into them. These intrusions were subsequently enriched from the same source by iron-bearing solutions or vapors, which "made the ore lenses that now comprise the ore bodies." Bayley makes no exception of the titanium-bearing deposits, and it seems very probable that they belong to this class, and not to the true "titaniferous magnetites" discussed in this report.

It is quite probable that a few of the high titanium occurrences may be exceptions and do occur in metamorphosed gabbros. Thus the Church or Van Syckle mine in Hunterdon County shows in a series of analyses a TiO_2 content averaging about 12 per cent, which is well within the limits of such ores.

THE TITANIUM-BEARING ORES.

The table following gives the analyses of those magnetites that contain more than 1 per cent TiO_2. They are derived from various sources, but are compiled in a series of tables in Bayley's report, the original source being given in each case.

[a] See p. 138.

Results of analyses of New Jersey ores.

Name of mine.	SiO_2.	Al_2O_3.	Fe_2O_3.	FeO.	MgO.	CaO.	Na_2O.	K_2O.	H_2O.	TiO_2.	P_2O_5.	S.	Cr_2O_3.	MnO.	V_2O_3.	Insol.	Total.	Fe.
Barker										1.47	Trace.	0.00		0.00				61.94
Bayard										6.00	Trace.							49.50
Beers										7.83	0.09	.31		.00				54.46
Bishop										1.98	.21	.39		.00				65.87
Bloom										4.77	Trace.	Trace.						37.5
Cannon [a]										2.77	Trace.							63.53
Charlotteburgh										1.02	Trace.	.39						64.94
Cramer I.										9.95	.32	.00		Trace.				62.23
Cramer II.										4.27	.90	(?)		Trace.				40.25
Day										5.08	.19	Trace.		.46				50.88
DeKay										4.68	.25	.41						45.75
Elizabeth [b]	1.38	0.55	65.26	30.20	0.10	0.68	Trace.	Trace.	0.12	1.09	.49	.01		.03	0.08		99.99	69.24
Do. [c]	8.46	.77	56.18	27.81	1.88	1.68	0.22	0.36	.28	1.06	.98	.01	0.00	.04	.11		99.84	61.02
Hager										1.40	.67	7.59		.00				56.13
Do. [d]										7.13	.44							56.39
Haggerty										4.27	.76	.05						58.55
Hann										1.07	.84	.09		.00				56.97
Hibernia [b]	12.08	1.63	56.21	26.39	.88	1.08	.17	.15	.17	1.21	.02	.03	Some.	.07	.21			59.93
Do. [c]	7.14	1.23	60.89	25.94	1.40	1.31	.08	.13	.25	1.25	.53	.01	.00	.03	.15			62.85
Holt Farm	18.90	12.48	65.52		.86	4.45				1.30	.06	1.23		.55				47.44
Horton	16.15	3.34	68.48		1.94	4.87				1.09	3.01						99.28	49.58
Hurd [b]	1.51	.37	64.87	31.11	.40	.51	Trace.			1.05	.05	.01	.00	.06	.13		100.33	69.69
Do. [c]	6.37	1.59	57.46	27.23	1.01	2.93	.12			.70	2.13	.05	.00	.03	.08		100.14	61.46
Jackson	9.80	9.48	73.15		.72	1.46				4.40	.54	.06					99.61	52.96
Kahart										1.42	.89	1.23						52.34
Keeler		1.17	75.95		Trace.	4.03				1.44	3.39	.01		.25		12.60	98.84	54.89
Leonard [b]	3.56	.44	61.47	55.24	1.68	1.66	.10			1.15	.54	.02	.00	.03	.08		100.05	65.88
Do. [c]	7.62	.49	29.04	26.12	1.89	4.34	.27			1.05	2.44	.02	.00	.04	.08		100.01	59.04
Marsh										2.08	.38	.06		.00				57.62
Martin										1.02	.58	.00						49.25
Naughright [e]			82.78		.91	2.70				6.40	2.56	.41				2.40	97.79	59.93
Do. [f]										7.62	.16	.27						64.77
Old Blue		.66	93.10		.43					1.95	.06					4.10	100.30	67.40
Richard [b g]	1.31	.73	64.01	30.62	.33	.96	Trace.	Trace.	.10	1.29	.56	.12	.00	.05	.10		100.18	68.69
Richard [e]	3.77	.79	61.16	29.56	.64	1.23	.12	.14	.16	1.30	.45	.01	.00	.06	.10		99.48	65.87
Do. [e]	8.48	.86	55.99	26.98	1.89	2.42	.33	.19	.15	1.01	1.54	.01	.00	.02	.08		99.95	60.34
Shafer										5.03	.21	.03		Trace.				45.22
Stockholm										3.65	Trace.	.61		4.86				67.10
Stony Brook										1.52	Trace.	.00		.00				64.69
Van Syckle	5.40	9.29	69.34		3.35	.39				11.60	Trace.	1.21		.00				50.20

[a] Three other analyses showed less than 1 per cent TiO_2.
[b] Selected sample. Tests for NiO, CoO, BaO, and SrO showed the absence of these constituents.
[c] Average sample. Tests for NiO, CoO, BaO, and SrO showed the absence of these constituents.
[d] Other analyses show 4.15 and 5.73 per cent of TiO_2.
[e] Gray ore.
[f] Blue ore.
[g] Another sample contained 0.30 per cent of TiO_2.

Bayley's report also includes 13 complete analyses of the ores in which V_2O_3 has been determined. These are of interest in comparison with the relation between the ratio of TiO_2 to V_2O_5 in certain Canadian ores and Adirondack ores as developed by Pope.[a] Pope found that a number of nontitaniferous magnetites which he tested were free of vanadium, whereas the titaniferous magnetites always showed some vanadium. Moreover, the ratio of vanadic oxide (V_2O_5) to titanic oxide (TiO_2) in the titaniferous magnetites from eastern Ontario and the Adirondacks was always in the neighborhood of 1 to 28. Consequently he suggested that in the titaniferous magnetites these two elements, vanadium and titanium, may be related to each other in a manner analogous to the relation of the oxides in definite ratios in the complex inorganic oxides studied by Gibbs, Marignac, and others.

The following table gives the percentages of TiO_2 and V_2O_3 determined in the New Jersey ores, and the ratio of TiO_2 to V_2O_5 calculated from them:

Vanadic oxide and titanic oxide in the New Jersey ores.

Mine.	TiO_2.	V_2O_3.	$TiO_2 + V_2O_5$.
	Per cent.	*Per cent.*	*Per cent.*
Hibernia	0.54	0.14	3.2
Richard	1.29	.10	10.7
Do	.30	.11	2.3
Do	1.30	.095	11.3
Do	1.01	.08	10.4
Hibernia	1.21	.21	4.7
Do	1.25	.15	6.9
Leonard	1.15	.08	11.9
Do	1.05	.08	10.18
Elizabeth	1.09	.08	11.2
Do	1.06	.11	8.0
Hurd	1.05	.13	6.7
Do	.70	.08	7.2

It is apparent from the table that not only is the proportion of vanadic oxide compared to the titanic oxide much greater in the New Jersey ores than in the ores studied by Pope, but also there is a wide variation from constancy in the ratio. The New Jersey ores can not be regarded as a fair test of the ratio, however, for, as already pointed out, they do not belong to the true "titaniferous magnetites." On the other hand, if they are thrown out of the class of titaniferous magnetites, the analyses would indicate that a vanadium content is not characteristic of the titaniferous ores alone.

NORTH CAROLINA.

Titaniferous magnetites occur at a number of localities in the Piedmont and Appalachian regions of North Carolina, but no deposits of great promise are known. The rocks of this western half of the

[a] Pope, F. J., Investigation of magnetic iron ores from eastern Ontario: Trans. Am. Inst. Min. Eng., vol. 29, 1899, pp. 395–397.

State consist principally of a complex of gneisses, ranging from acidic or granitic gneisses to very basic rocks, as hornblende gneisses, granites, and other igneous rocks. The most recent and complete account of the titaniferous magnetites occurring in this region is contained in the bulletin on the iron ores of the State by Nitze.[a] This report includes all information in previous reports of the State survey, and, except for data relative to the localities visited by the author of this report, it forms the basis of the following account.

As most of the localities cited in the North Carolina reports show no evidence of the existence of deposits of any importance, it was considered useless to attempt to visit all, and at the suggestion of Dr. J. H. Pratt, State geologist of North Carolina, the author visited what were considered the two most promising areas, the belt extending across Rockingham, Guilford, and Davidson Counties, just west of Greensboro, and the area to the north of Lenoir, in Caldwell County.

The general distribution and character of the ores is very briefly indicated; a more detailed description is given only of the two areas visited by the writer.

DISTRIBUTION AND CHARACTER OF THE ORES.

References to the character of the immediate country rock of the ores are very meager and of a most general nature. All that can be gotten from the literature is that it is in most cases a schistose femic rock. No information as to what the original rock might have been is given. Little can be said, therefore, in regard to the geology of the ores.

Our knowledge of the character and extent of the ores is more complete, as a large number of analyses have been made, and some data are given indicating the size of the ore bodies. The table following gives the available analyses, arranged alphabetically by counties. It is the same as that given by Nitze,[b] with five analyses added in Guilford County taken from Lesley.[c]

[a] Nitze, H. B. C., A preliminary report on the iron ores of North Carolina: North Carolina Geol. Survey Bull. 1, 1893.

[b] Op. cit., pp. 229–231.

[c] Lesley, J. P., Notes on the titaniferous iron belt near Greensboro, North Carolina: Proc. Am. Phil. Soc., vol. 12, 1871, pp. 154–156.

Results of analyses of North Carolina ores.

Source of ore.	SiO_2.	Fe.	Mn.	S.	P.	TiO_2.	Cr_2O_3.	Al_2O_3.	$CaCO_3$.	$MgCO_3$.	CaO.	MgO.
ALLEGHANY COUNTY.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1. Fielden Carrico, Old Forge workings	6.20	54.72	.....	0.038	0.047	4.86	.....	.....	.....	.....	.....	.....
ASHE COUNTY.												
2. Cicero Pennington, Wallens Creek	4.75	52.23	.....	.112	.021	8.91	1.19	.....	.....	.....	.....	.....
3. Cicero Pennington, Wallens Creek	4.72	52.44	.....	.077	.004	5.38	.....	.....	.....	.....	.....	.....
4. Cicero Pennington, Wallens Creek	5.07	52.45	.....	.....	.022	9.11	.....	.....	.....	.....	.....	.....
5. Bauguess opening, near Little Helton Creek	6.35	57.66	.....	.061	.008	4.69	.505	.....	.....	.....	.....	.....
6. Bauguess opening, near Little Helton Creek	7.91	53.35	.....	.078	.022	4.92	.....	.....	.....	.....	.....	.....
7. G. C. McCarter, Little Helton Creek	9.90	46.81	.....	.137	.025	6.03	.63	.....	.....	.....	.....	.....
8. G. C. McCarter, Little Helton Creek	10.92	40.71	.....	.065	.012	5.54	.....	.....	.....	.....	.....	.....
9. G. C. McCarter, Shipps Branch opening	5.37	51.75	.....	.....	.018	9.17	.....	.....	.....	.....	.....	.....
10. William Young, Little Helton Creek	5.12	50.77	.....	.04	.005	4.95	.....	.....	.....	.....	.....	.....
11. William Young, Little Helton Creek	4.35	52.85	.....	.....	.013	8.80	.....	.....	.....	.....	.....	.....
CALDWELL COUNTY.												
12. J. K. Farthing, Warrior Creek	6.50	31.92	0.39	.058	.025	2.40	.....	.....	7.48	15.64	.....	.....
13. Joshua Curtis, Yadkin River (average)	6.63	36.00	1.09	.021	.060	14.90	.....	.....	7.37	16.08	.....	.....
14. Joshua Curtis, Yadkin River (selected)	7.55	28.24	.....	.013	.140	41.21	.....	.....	.....	.....	.....	.....
15. Joshua Curtis, Yadkin River	.....	37.10	.....	.....	Trace.	36.40	.....	.....	.....	.....	.....	.....
16. Joshua Curtis, Yadkin River	.....	25.76	.....	.....	.076	38.81	.....	.....	.....	.....	.....	.....
CATAWBA COUNTY.												
17. Forney ore bank, near Maiden Station	1.41	67.92	.....	.07	.025	1.60	.....	.....	.....	.....	.....	.....
DAVIDSON COUNTY.												
18. K. R. Swain, massive ore	.76	57.68	.....	.....	.....	13.52	.46	1.68	.....	.....	.....	.....
19. K. R. Swain, micaceous ore	5.68	52.68	.....	.....	.....	11.67	.48	5.08	.....	.....	.....	.....
DAVIE COUNTY.												
20. Kelley-Lefler place, 5 miles south of Mocksville	.778	60.00	.....	.033	.008	10.32	.....	.....	.....	.....	.....	.....
21. Allen place, 7 miles northeast of Mocksville	5.50	52.80	.....	.11	.02	8.00	.....	.....	.....	.....	.....	.....
GUILFORD COUNTY.												
22. Elisha Charles	.40	59.03	.....	.....	.....	11.95	1.07	1.06	.....	.....	.....	.....
23. Widow Cook	1.84	56.21	.....	.....	.....	13.28	.65	2.30	.....	.....	.....	.....
24. John Clark	1.30	56.41	.....	.....	.....	12.35	1.10	2.54	.....	.....	.....	.....
25. Sargeant shaft	1.31	55.06	.....	.....	.....	13.60	.72	4.26	.....	.....	.....	.....
26. Sargeant shaft	.....	53.20	.....	.....	.005	Present.	.....	.....	.....	.....	.....	.....

27. Sargeant shaft	12.86	53.27	.....	.....	.....	13.58	.....	.....	.....	.....	.....	.....
28. Mrs. McCuiston, soft micaceous	.....	43.47	.....	.....	.....	16.06	.....	.....	.....	.....	.....	.....
29. Mrs. McCuiston, soft micaceous	12.75	41.95	.....	.....	.....	15.35	1.25	5.17	.....	.....	.....	.....
30. Mrs. McCuiston, magnetic part of above	1.30	67.60	.....	.....	.....	1.27	1.43	.55	.....	.....	.....	.....
31. Mrs. McCuiston, nonmagnetic part of above	26.80	21.63	.....	.....	.....	16.20	.43	8.87	.....	.....	.....	.....
32. Mrs. McCuiston, massive ore	.....	33.97	.....	.....	.....	2.63	.....	.....	.....	.....	.....	.....
33. W. A. Lewis	.50	57.32	.....	.....	.....	12.27	.57	3.62	.....	.....	.....	.....
34. Belt west of Greensboro *	.75	57.77	.22	.....	.....	12.08	.32	4.62	.....	.....	0.13	2.04
35. Belt west of Greensboro	1.89	59.95	.32	.....	.....	8.72	.40	3.93	.....	.....	.17	1.36
36. Belt west of Greensboro	1.04	58.52	Trace.	Trace.	.00	12.32	Trace.	3.87	.....	.....	.64	.49
37. Belt west of Greensboro	1.50	60.41	.08	.00	Trace.	8.65	.83	2.90	.....	.....	.75	2.02
38. Belt west of Greensboro	.52	57.30	.48	.00	.00	13.74	Trace.	4.50	.....	.....	.72	.54
LINCOLN COUNTY.												
39. Lawson Bess, Indian Creek	11.37	56.95	.....	.045	.029	2.40	.....	.....	.....	.....	.....	.....
MACON COUNTY.												
40. Felix Kilpatrick, 5 miles east of Franklin	.77	54.24	.....	.04	.013	17.60	.....	.....	.....	.....	.....	.....
41. Alex. Waldroop, 6 miles west of Franklin	11.91	20.64	.69	.069	.016	3.20	.....	.....	3.31	14.69	.....	.....
MADISON COUNTY.												
42. Road, midway between Asheville and Burnsville	.83	36.26	.63	.09	Trace.	37.88	.....	9.51	.....	.....	.....	.....
43. John Brigman, Paint Fork	Titanic acid present in large quantity.											
44. Swan Woody, Spring Creek	2.37	62.16	.....	.026	.014	7.44	.....	.....	.....	.....	.....	.....
MITCHELL COUNTY.												
45. Avery land, Roaring Creek, Yellow Mountain	1.46	65.32	.....	.025	.009	4.80	.....	.....	.....	.....	.....	.....
46. Avery land, Roaring Creek, Yellow Mountain	.54	66.95	.....	None.	.015	6.80	.....	.....	.....	.....	.....	.....
47. Parker place, 2 miles west of Burnsville	1.22	57.98	.....	.015	.041	4.56	.....	.....	.....	.....	.....	.....
48. Joel Gouge, mouth of Little Rock Creek	1.13	64.56	.....	.027	.078	4.48	.....	.....	.....	.....	.....	.....
49. Above Jenkins mine, Greasy Creek	6.58	54.48	.....	.023	.033	4.96	.....	.....	.....	.....	.....	.....
ROCKINGHAM COUNTY.												
50. Levi Shaw	1.80	54.17	.96	.....	.....	14.46	.97	2.66	.....	.....	.....	.....
51. P. Hopkins	.74	55.61	.82	.....	.....	13.92	1.07	3.82	.....	.....	.....	.....
52. Granular reddish ore	1.39	30.97	.63	.....	.....	.78	.30	52.24	.....	.....	.....	.....
53. Granular grayish ore	.98	33.52	.80	.....	.....	2.42	Trace.	44.86	.....	.....	.....	.....
54. Dannemora mine	4.71	48.41	.....	.089	.023	13.74	.34	8.68	.....	.....	.....	.....
55. Dannemora mine, fine ore	.....	49.41	Trace.	.....	.001	Present.	.....	.....	.....	.....	.....	.....
YANCEY COUNTY.												
56. Mine Fork, 6 miles north of Burnsville	9.25	39.42	.....	.12	.011	11.90	.....	.....	.....	.....	.....	.....
57. Jerry Ferguson, 9 miles west of Burnsville	23.38	39.00	.....	.....	.....	2.56	.....	.....	.....	.....	.....	.....

* Ore represented by this analysis had a trace of V_2O_5.

These analyses show a considerable range in TiO_3 content; whereas a number carry only a small percentage of TiO_2, others have such a high percentage that the ore must consist almost entirely of ilmenite. The phosphorus ratio in nearly all of the ores is below the Bessemer limit, and sulphur is low. The percentage of iron is also high in most of the analyses. Except for their titanium content, most of the ores would be classed as medium to high grade.

ALLEGHANY COUNTY.

A zone of hornblende schists, in many places altered to steatite or soapstone, and carrying crystalline grains of magnetite, usually titaniferous, extends across Alleghany County parallel to the Little River and usually on the south side of it. Locally the magnetite becomes more abundant in the schists and may be sufficiently concentrated to form small lenses of ore. On the farm of Fielden Carrico, near the State line, several such ore bodies were opened up about 30 years ago to supply a near-by forge in Virginia (see analysis 1). These openings have long since been abandoned and caved in.

ASHE COUNTY.

A belt of titaniferous magnetite extends in a northeasterly direction from Helton Creek near Sturgill to Little Helton Creek, a distance of 2½ miles. The analyses of the ores in this belt (analyses 2 to 10) show a moderate titanium content.

At the northeast end of the belt is a series of outcrops, one of which shows an ore bed at least 25 feet wide. It is a coarse granular magnetite practically free from gangue. The country rock is in part a prophyllite schist and in part a hornblende rock altering to asbestos. South of these outcrops is another showing 5 feet of ore with a gangue of epidote, feldspar, and quartz. At the southwest end of the belt are several more outcrops, in which the width of the ore ranges from 8 to 15 feet. The ore is a fine-grained magnetite with a gangue of hornblende and epidote.

CALDWELL COUNTY.

Two localities in this county were visited by the writer, but neither showed either large or promising deposits.

RICHLANDS COVE DEPOSIT.

Sixteen miles north of Lenoir in what is known as the Richlands cove is a body of titaniferous iron ore on the east bank of the Yadkin River. A small opening has been made adjacent to the river bank exposing a face about 20 feet high and 40 feet wide. This titaniferous mass occurs within a country rock consisting of sericitic schist. It consists of small particles of ore in a matrix chiefly made

up of fibrous and scaly aggregates of chlorite, serpentine, and talc. The individual ore particles average less than one-half mm. in diameter, and rarely exceed 1 mm. They are very slightly magnetic to nonmagnetic. Two polished sections of the ore etched with hydrochloric acid retained their luster and showed no evidence of the intergrowths of ilmenite and magnetite. These facts, together with the high titanium content shown in the analyses in the table (analyses 13 to 16), indicate that the ore particles consist chiefly of ilmenite. This is further borne out by the fact that the sands along the river close to the deposits contain only nonmagnetic ore particles.

Just north of the opening on the river bank is a small side valley in which a rock of identical character outcrops. This outcrop is about 300 feet from the river on the north side of the valley.

Neither of these deposits shows extensive outcrops, and they seem to be only small intercalations in the more acidic country rock. They undoubtedly represent small basic intrusions that have undergone metamorphism with the rest of the region.

The titanium content is so high that the deposits can not be regarded as iron ores. As sources of titanium, their inaccessible location and seeming small size are great drawbacks. At present they possess no economic value.

FARTHING DEPOSIT.

On the north side of Warrior Creek, 5½ miles north of Lenoir, there are outcrops of an ore-bearing rock. The rock consists principally of a green hornblende schist carrying considerable ore disseminated in minute grains and in stringers of nearly pure ore. It occurs as an intercalation in more acidic schistose rocks. Similar rock can be traced for some distance to the southwest by means of surface fragments, but no openings have been made on this occurrence.

Locally this hornblende rock contains richer portions of fine-grained ore. A thin section of such a piece consisted principally of the ore minerals and spinel. Less abundant were augite and hornblende. The ore grains are less than 0.5 mm. in diameter, and do not constitute more than one-third of the mass. On etching polished surfaces of these ores, the small magnetite grains are dissolved out without showing any ilmenite intergrowths, and the polished surfaces of the ilmenite remain unattacked. Further evidence that the small pits on the polished surface were occupied by magnetite is the fact that a considerable part of the powdered ore is magnetic. Analysis 12 shows the composition of the ore.

Adequate exposures to enable one to judge accurately the extent of the occurrence are lacking. The richer parts of the rock are so lean, however, as to preclude any possibility of working the deposit.

CATAWBA COUNTY.

A belt of titaniferous ore extends across parts of Catawba and Lincoln Counties, from a point southwest of Newton to about 9 miles west of Lincolnton. At the northeast end is the Forney mine, which formerly supplied several forges in the neighborhood. The ore is a coarse granular magnetite, usually free from gangue, and occurs in irregular pockets a few inches to 3 or 4 feet in thickness. The country rock is called syenite in the North Carolina report. An analysis of the ore (analysis 17) shows a low titanium content. Float ore found in this belt at the southwest end also shows (analysis 39) only a small percentage of titanium.

DAVIDSON COUNTY.

See Guilford County.

DAVIE COUNTY.

Titaniferous ores are described at two points in Davie County near Mocksville.

About 5 miles south of Mocksville, near the mouth of Bear Creek, is a fair showing of float ore for a distance of 600 feet on the old Maxwell place. Float ore can be traced at intervals for 1½ miles to the southwest to the South Yadkin River. The country rock is described as "hornblende and syenite, with occasional dissemination of magnetite granules." The ore is a medium grain magnetite with little gangue (analysis 20).

Similar ore occurs 7 miles northeast of Mocksville and 1 mile east of Kernatzer station. Several pits and a shaft were put down here during the Civil War. The ore was worked in a forge 4 miles distant on Dutchmen's Creek. Float ore can be traced faintly 200 to 300 yards northeast and southwest of the old shaft (analysis 21).

GUILFORD COUNTY.

A belt of titaniferous magnetite extends across Guilford County in a northeasterly direction, and passes beyond the boundaries of Guilford County into Rockingham County on the northeast and into Davidson County on the southwest. It lies to the north and west of Greensboro. It consists of two parallel subordinate belts known, respectively, as the Tuscarora belt and the Shaw belt, the total length of which is 30 miles. The longer and more persistent is the Tuscarora belt. The Shaw belt lies several miles northwest of it.

The area is one of prevailing granites and gneisses within which are smaller bodies of gabbro. Though evidence of the presence of the gabbro is obtained repeatedly along the belt by outcrops and surface fragments, it seems that the belt is not one elongated mass of gabbro, but consists of a number of smaller masses having a linear distribution along the belt. The ore bodies are segregations within

these small gabbro masses and do not constitute a continuous body extending the entire length of the belt.

Surface outcrops of the ore bodies are lacking, and as the old workings have all caved in the character of the ore can be judged only from the surface fragments. A detailed examination of the belt was made by Lesley[a] for a Philadelphia mining company, and trenching was done at a number of points to expose the ore. Dr. Lesley consequently had unusual opportunities for observing the ore bodies, and his account of them is the most valuable we have. Shortly after this the belt was visited by Prof. W. C. Kerr and Dr. F. A. Genth, and described in a report of the North Carolina geological survey.[b] The descriptive part is taken mainly from Lesley's report, but 16 additional analyses made by Genth are given. This same report was reprinted with slight alterations in 1883[c] and again in 1893.[d] A brief sketch is also given in the Tenth Census reports.[e]

It is not surprising that Dr. Lesley at that early date misinterpreted the genesis of the ores. He regarded them as bedded deposits of sedimentary origin, and consequently the conclusions he drew as to the geology and structure of the deposits are erroneous. These are of historical interest only and will not be repeated here. The records of his observations, however, supply facts no longer accessible on account of the filling of the old workings and trenches. The ore bodies consist of strings of lens-shaped masses, continually enlarging and contracting in thickness from a few inches to 6 and 8 feet. "The principal beds may be safely estimated at an average of 4 feet, and in the best mining localities the average yield of a long gangway may reach 5 feet." The occurrence of the ore body in a series of disconnected lenses is consequently verified by the observation of Prof. Lesley.

TUSCARORA MINE.

This mine was worked in the seventies by a Philadelphia mining company. Forges were erected on a small stream a half mile northeast of the principal shaft, and considerable iron is said to have been made. It seems that the titaniferous ores could readily be smelted in the old Catalan forges and yielded a superior grade of iron. The Tuscarora mine consisted of several shafts extending about a mile along the ore belt. The principal shaft, the Sargeant, was situated a mile and a half north of Friendship. It reached a depth of 109 feet, and a tunnel run in from this depth is said to have cut a bed 12 feet wide.

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[a] Lesley, J. P., Note on the titaniferous iron-ore belt near Greensboro, N. C.: Proc. Am. Phil. Soc., vol. 12, 1871, pp. 139–156.

[b] Report of the Geological Survey of North Carolina, vol. 1, 1875, pp. 236–250.

[c] Kerr, W. C., and Hanna, G. B., Ores of North Carolina: Geol. Survey of North Carolina, 1888, pp. 143–154.

[d] Nitze, H. B. C., A preliminary report on the iron ores of North Carolina: North Carolina Geol. Survey Bull. 1, 1893, pp. 60–66.

[e] Tenth Census, vol. 15, 1886, pp. 308–311.

Outcrops of granitic gneisses occur in the vicinity of the mine, but the immediate country rock is a gabbro. It is an olivine gabbro with diabasic texture, in which the feldspar laths reach a length of 2½ mm., and the pyroxene and olivine grains have a diameter of 1 to 1½ mm.

No ore in place is now exposed, but judging from fragments lying about on the surface it is a medium to coarse grained ore of good grade, in which the principal gangue consists of tufts and seams of chlorite and small quantities of material so decomposed as to make it undeterminable. Analyses 25 to 27 give the composition of the ore.

METALLOGRAPHIC DESCRIPTION OF ORE.

Polished sections of this ore show very striking features. Gangue minerals constitute only a small part of the surface. Ilmenite grains make up one-fifth to two-fifths of the surface; and they range in size from 2 mm. to 0.2 mm., though very few fall below 0.5 mm. in diameter. The most striking feature of the ore is the ilmenite intergrowths in the magnetite, which attain a coarseness not approached in any of the other ores that have been studied. Indeed, so coarse are they as to be easily discernible with the naked eye. An etched specimen of the ore is shown in Plate II, *C*, in which the relations of the ilmenite and the magnetite grains and the heavy ilmenite laths in the magnetite are apparent. The individual ilmenite plates have an average length of 2 mm., but some attain a length of as much as 4 mm. The space between the parallel plates varies from 0.2 to 0.5 mm., and the plates themselves may be as thick as 0.1 mm. Another characteristic feature is protuberances or local thickenings of the plates. This most frequently takes place at one end, the other end thinning out, giving an elongated, wedge-shaped appearance. These plates are plainly visible on cleavage faces of the magnetites on the unpolished surfaces of pieces that have been etched, and are frequently coarse enough to peel off with the edge of a knife blade. The structure is shown in Plate VII, *A*.

POSSIBILITIES OF UTILIZING ORE.

The most satisfactory results of magnetic separation were obtained from this ore, as shown below:

Results of magnetic separation of Tuscarora ore.

	Unscreened ore.	Ore through 50-mesh, over 100-mesh.[a]		Ore through screen finer than 100-mesh.[b]
		Magnetic.	Nonmagnetic.	Magnetic.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent*
Quantity	.....	71.5	28.5	70.1
Fe	58.07	67.76	33.76	68.41
TiO_2	12.82	4.25	34.32	3.64

[a] 44.6 per cent.

[b] 55.4 per cent.

A. TUSCARORA MINE, N. C., ORE. (× 60.)

B. IRON LAKE, MINN., ORE. (× 60.)

These results show that little is to be gained by grinding finer than 50-mesh; that is, notwithstanding the coarseness of the ilmenite intergrowths they are still too fine to be separated by grinding to 100-mesh. This of course is obvious from a comparison of the dimensions of the intergrowths given above and the size of the openings of the 100-mesh screen. If all the TiO_2 in the tailings is assigned to the ilmenite, they contain about 75 per cent ilmenite, with a residue of 6.38 per cent Fe. Some of this iron occurs in the gangue minerals present, so that very little magnetite is lost by adhering to the ilmenite or gangue. Although the titanium content of the concentrate is still higher than is acceptable in present blast-furnace practice, the concentrate is so high grade that it would make an acceptable ore to mix with titanium-free ore.

Kerr [a] gives the results of a magnetic-separation test made with a common magnet on ore obtained from this same belt several miles northeast of the Tuscarora mine. They are even more favorable than those presented above.

Results of magnetic-separation test of Tuscarora ore.

Constituent.	Natural ore.	Magnetic.	Nonmagnetic.
	Per cent.	*Per cent.*	*Per cent.*
Fe	41.95	67.60	21.63
TiO_2	15.35	1.27	16.20
SiO_2	12.75	1.30	26.80

TRUEBLOOD PLANTATION ORE.

About 2 miles northeast of the Tuscarora mine, on the south side of Brushy Creek, a little work has been done. The ore here also occurs in association with a gabbro. The gabbro is strongly uralitized and the feldspar sufficiently altered to make the twinning obscure in places. Diabasic texture is far less prominent than in the rock of the Tuscarora mine, and often lacking entirely. The average size of the grains is about 2 mm.

The ore is not quite as coarse as the Tuscarora mine ore, and etched sections do not show the intergrowths as abundantly. Otherwise the ore at the two localities is very similar in appearance.

APPLE PLANTATION ORE.

Some pits were put down on the Apple plantation just over the line in Rockingham County, southeast of the Haw River, but they are all filled now. Fragments of ore found on the surface are similar to the ore found on the Trueblood plantation. Of special interest to this locality are fragments of pink and gray emery. The occurrence

[a] Report of the Geological Survey of North Carolina, vol. 1, 1875, p. 245.

of emery in this belt is first mentioned in Kerr's report,[a] in which he gives two analyses made by Genth. One analysis is of a reddish granular ore (analysis 52), which he describes as follows: "It has much the appearance of a granular reddish-brown garnet, for which it has been mistaken, until the analysis proved it to be not a silicate mixed with granular magnetite, but corundum." The other is of a granular grayish ore (analysis 53) described as follows: "This is of a similar quality and is found at the same locality; the minute grains of corundum have a yellowish or brownish-white color, and show in many places cleavage fractures, which give it the appearance of a felspathic mineral." No mention is made of the locality where these specimens were found. Pratt and Lewis[b] state that the exact location is not now known, and give the McCarviston or McCuiston place, 7 miles northeast of Friendship, in Guilford County, as the probable location. The specimens found on the old Apple plantation fit the description given by Kerr exactly, and as no such material was found anywhere else in the belt, it seems probable that this is the locality from which Kerr's specimens were obtained. The rock was not found in place and no conclusions could be drawn as to the occurrence and size of the deposit.

DANNEMORA MINE.

Some work was done on the Shaw belt, 3 miles west of Summerfield, but the only extensive workings were those on the north side of the Haw River, close to the line between Guilford and Rockingham Counties. Ore was worked from pits in Revolutionary times, and hauled to Troublesome Forge, 5 miles to the north. Subsequently a Philadelphia mining company worked the Dannemora mine on this property. The following description of the mine, given by Bailey Willis in the Tenth Census report,[c] is quoted because it gives the size of one of the ore lenses:

> The first work was done in incline No. 1. According to the statement of the superintendent the ore gave out 10 feet from the surface but was found again 20 feet farther down; it widened to 12 feet, which is the thickness 90 feet from the surface in the incline, 70 feet vertically. At this depth a drift has been driven along the vein and stoping has been begun. A small winze has been sunk 20 feet farther on the incline, and the ore is said to narrow to 1 foot in thickness. Before the horizontal drift reached the second incline the ore was pinched out by the granite walls.
>
> As the ore body thus opened gives out just beyond incline No. 1, its dimensions are pretty well ascertained. It is approximately 125 feet long, 80 feet in incline width, and 12 feet thick. The drift has been extended to another lens of the same thickness, northeast of it, and the surface outcrops and trenches indicate the existence of others to the southwest. The ore is accompanied by chlorite and mica, which are sufficiently

[a] Report of the Geological Survey of North Carolina, vol. 1, 1875, p. 246.

[b] Pratt, J. H., and Lewis, J. V., Corundum and the peridotites of western North Carolina: North Carolina Geol. Survey, vol. 1, 1905, pp. 263–264.

[c] Tenth Census, vol. 15, 1886, p. 308.

decomposed, even at the depth of the tunnel, to be readily dug with a pick. The ore is separated on the surface, by screening, into lump and fine ore, the former being shipped for fettling, the latter for blast-furnace use.

The ore lying about the old workings is somewhat leaner than the Tuscarora-mine ore. It consists of a similar granular aggregate of ilmenite and magnetite with the gangue minerals, but the individual grains seldom exceed 1 mm. in diameter, and most of them are about 0.5 mm. Intergrowths of ilmenite in the magnetite grains are almost lacking.

SUMMARY.

The titanium content of these ores is about the normal one for titaniferous magnetites. Of the large number of available analyses the average TiO_2 content is about 13 per cent. The ilmenite and the magnetite occur as granular aggregates in the ores in such a manner that most of the titanium can be eliminated with little loss of magnetite. The resulting concentrate runs high in iron and is low enough in titanium to make possible the utilization of these ores in normal blast-furnace practice by mixing them with titanium-free ores. On the other hand, little is known in regard to their geologic occurrence and size on account of lack of exposures. Such conclusions as can be drawn from old data and the nature of the deposits are not very favorable to them. The ore bodies seem to be rather small and irregular in distribution, so that mining operations would be attended by considerable uncertainty. On the whole, therefore, the belt is not very promising as a source of iron ore.

LINCOLN COUNTY.

See Catawba County.

MACON COUNTY.

There are a number of occurrences of titaniferous magnetite in Macon County east and south of Franklin, but none has been explored more than superficially.

Five miles east of Franklin and one-eighth of a mile north of Culasagee Creek there is a heavy surface float of massive lustrous magnetite high in titanium (analysis 40). The gangue is quartz and chlorite; the ore is traceable for several hundred yards along the summit of a hill in a northeasterly direction parallel to the strike of the decomposed mica schists near the foot of the hill.

An analysis has also been made of the ore on land 4 miles southwest of Franklin and 2 miles above the mouth of Cartoogajay Creek (analysis 41). The ore is very fine grained, in a chloritic gangue, and in places slightly garnetiferous. The extent of the ore body is not apparent.

MADISON COUNTY.

No important bodies of titaniferous ore are known in Madison County, though there are several occurrences of the ore.

In the eastern part of the county, a half mile above the mouth of Paint Fork, small pieces of float ore high in titanium are found (analysis 43). The occurrence of a highly titaniferous ore is also noted near the road midway between Asheville and Burnsville (analysis 42).

An occurrence showing only a moderate amount of titanium is that a half mile east of the Haywood County line near the headwaters of Spring Creek (analysis 44). It has a thickness of 5 to 6 feet near the surface and appears to widen in depth.

MITCHELL COUNTY.

Titaniferous ores occur in Mitchell County in the Roan Mountain belt, the Pumpkin Patch Mountain belt, and on Iron Mountain.

Titaniferous ore occurs on one of the spurs of Iron Mountain 2½ miles above the mouth of Greasy Creek, near the Jenkins ore bank (analysis 49). The ore body consists of a compact lustrous ore free from gangue, and attains a width of 5½ feet. The wall rock is hornblende gneiss and pegmatite.

The Roan Mountain titaniferous belt lies several miles south of the Cranberry belt to which it is parallel. Its eastern extremity is near the mouth of Roaring Creek on one of the southern spurs of Big Yellow Mountain. It crosses the State line on the summit of Grassy Bald Ridge, then, after traversing the eastern edge of Tennessee for about 4 miles, bends around to the southwest and reenters North Carolina near the headwaters of Big Rock Creek; thence it continues to the mouth of Big Rock Creek to the Yancey County line at Toe River. No ore bodies of any size have been found along this belt. At the eastern end some prospecting has been done, including the drilling of three diamond-drill holes. The results were discouraging, as the ores occur in lenses of 2 inches to 2 feet in width. The titanium content is only moderate in amount (analyses 45 and 46). The ore is compact, lustrous, and free from gangue, and occurs in a country rock of very coarse-grained pegmatite, hornblende schist, epidote, and garnet rock.

At the southwest end of the belt on the north side of Little Rock Creek a small pit exposes a bed 3 feet wide, which at a depth of 4 feet is cut out by a diabase dike (analysis 48).

The Pumpkin Patch Mountain titaniferous magnetite belt lies to the northwest of Bakersville and has an extent of 5 to 6 miles. Some pits have been put down on the Parker place near the head of Wadkins Branch, 5 miles northwest of Bakersville, and loose blocks of

ore encountered (analysis 47); the pits did not go deep enough to get the ore in place.

All of these Mitchell County ores show approximately the same titanium content.

ROCKINGHAM COUNTY.

See Guilford County.

YANCEY COUNTY.

The occurrences of iron ore in Yancey County are isolated and sporadic, and no notable occurrences of titaniferous ore have been disclosed.

In the extreme western part of the county near the head of Possum Trot Creek, 9 miles west of Burnsville, is found float ore that shows a small amount of titanium (analysis 57). It is possible that this occurrence belongs to the Roan Mountain titaniferous belt of Mitchell County.

Another deposit occurs 6 miles north of Burnsville, on the south side of Mine Fork, a half mile above its mouth (see analysis 56). Two small openings were made 75 feet apart on the strike of the ore (N. 25° E.) and show a vertical bed of ore 6 to 10 feet across. The ore is in a gangue of chlorite, small particles of quartz and feldspar, and a peculiar brown mineral of high luster which may be rutile or brookite. A quarter mile southwest of these openings is much surface float of this heavy brown mineral nearly free from magnetite. If this mineral is really rutile or brookite, considerable titanium could easily be eliminated by magnetic concentration.

MINNESOTA.

Titaniferous magnetite occurs within an extensive area of gabbro in northeastern Minnesota known as the Duluth laccolith. This forms part of the pre-Cambrian region of Lake Superior, and the geology and petrography of it have been referred to repeatedly by the geologists who have worked in this field. The geology of the ore bodies is so simple and that of the entire area so complicated that it would be a needless use of space in this report to attempt to give a résumé of the geological literature embracing this area. A bibliography of the literature is given in Monograph 52 of the United States Geological Survey.[a]

Describing the areal distribution of the great gabbro mass Elftman[b] says:

> The gabbro forms the outer or northern member of the Keeweenawan series. The general shape of the area is crescentic with the concave side toward Lake Superior.

[a] Van Hise, C. R., and Leith, C. K., The geology of the Lake Superior region: U. S. Geol. Survey, 1911, pp. 78–80.

[b] Elftman, A. H., The geology of the Keeweenawan area in northeastern Minnesota: Am. Geol., vol. 22, 1898, p. 132.

Winchell,[a] referring to ore which he found on the north shore of Iron Lake, says: "This iron ore constitutes what is locally known as the 'Mayhew Iron Range,' and is found in a belt about a mile wide on both sides of Iron Lake, and on the south side of Portage Lake, and between Portage and Poplar Lakes. An analysis reported by Mr. Hause, performed by R. S. Robertson, of Pittsburgh, Pa., gave the following result: Silica, 2.02; alumina, 2.68; titanium, 12.09; sesqui chromium, 2.40; mag. ox. iron, 80.78; lime, trace; phos. acid, 0.03." A careful study of this location warranted among others the following statements:[b]

First. The ore is in the igneous rock.

Second. It varies in quality very much, even passing into rock that can not be styled iron ore.

Tenth. The iron in considerable but unknown quantity is of fine quality, being seen in other places besides this described.

From several assays Mr. Hoare states that he ascertained the iron and titanium together in the ore from Iron Lake to be pretty nearly uniform throughout, but that the titanium varied from 6 to 13 per cent, displacing about the same percentage of iron. The ore is in enormous quantities, in one place laid bare so that he crossed over it 30 paces.

A year later, in a report on the mineralogy of Minnesota, Winchell[c] said of menaccanite:

This seems to be the principal magnetic mineral which enters into the gabbro and other igneous rocks of the Cupriferous in Minnesota. It is so abundant in the region of Iron Lake, north of Grand Marais, and at Herman, near Duluth, that it has attracted attention as an iron ore. The titanium present is frequently not enough to constitute the mineral, which requires from 30 to 40 per cent. It is sometimes as low as 3 per cent, the mineral then being rather titaniferous magnetite.

Irving[d] mentions the same localities. In speaking of the orthoclase-bearing gabbros, he says:

In some of these rocks the titaniferous magnetite forms bunches of such size as to have attracted attention as an iron ore, as, for instance, at Duluth, and again at several points in the interior back of Grand Marais and Beaver Bay. From some of this ore obtained some miles back of Grand Marais I obtained, some years since, over 10 per cent of titanic acid.

In 1887, Winchell[e] mentioned the occurrence of titaniferous iron ore in abundance on the shore of Gabimichigama Lake. In a letter to the author dated January 28, 1912, Mr. John G. Howard, of Duluth, writes the following in regard to this same locality: "The ore marked from section 7–64–5 and section 12–64–6 comes from the county line between Cook and Lake, and crosses the line and outcrops in

[a] Winchell, N. H., Tenth Annual Report of the Geological and Natural History Survey of Minnesota, 1881, p. 81.

[b] Idem, pp. 80–81, 85.

[c] Winchell, N. H., Eleventh Annual Report of the Geological and Natural History Survey of Minnesota, 1882, p. 17.

[d] Irving, R. D., The copper-bearing rocks of Lake Superior: Mon. 5, U. S. Geol. Survey, 1883, pp. 51–52.

[e] Winchell, Alexander, Fifteenth Annual Report of the Geological and Natural History Survey of Minnesota, 1886, p. 167.

Lake County in section 12–64–6, and also in section 7–64–5, Cook County. This is a fine large outcrop and assays as follows: Iron, 56.16; phosphorus, 0.003; silica, 2.40; manganese, 0.40; lime, 0.04; magnesia, 1.35; sulphur, trace; titanium, from none to 6. This deposit shows on surface for over a mile in length and shows lots of clean ore." Mr. Howard also states that there are a number of deposits between this location and Iron Lake, which lies about 18 miles to the east. Another analysis furnished by Mr. Howard comes from a locality still farther west in section 30–63–9, and is as follows: Iron, 55.04; phosphorus, 0.003; silica, 6.20; lime, 0.15; manganese, 0.28; magnesia, 2.28; alûmina, 8.85; sulphur, 0.876; titanium, from 2.50 to 19.

Wadsworth,[a] in describing the Duluth gabbro, says: "Others, again, have the magnetite abundantly developed, so much so that the grayish-black rock has been designated an iron ore, which has been mined at Duluth." Again he says:[b] "In local modifications of gabbro, the magnetite predominates to such an extent over the other minerals that it is mined as an iron ore. The quantity of this ore is very great, and a number of mines exist upon it at various localities, * * * the only apparent limit to it being extensively mined is owing to the titaniferous nature of the ore. * * * These ores generally seem to be titaniferous magnetite, the titanium varying from a minute amount up to 12 per cent or higher."

Winchell again refers to Iron Lake in the sixteenth annual report of the geological and natural history survey of Minnesota. He says:[c]

> The rim of the lake is mostly massive gabbro. This in places is so strongly charged with magnetite that one might truthfully conceive of the prison lake as iron-bound. The abundance of the magnetite caused attention within a few years to be directed to the region, but investigation showed the ore to be strongly titaniferous.

Describing a point on the north shore, he says:[d]

> Black sheets of magnetite are seen spreading out in the gabbro in an irregular way. Here and there fragments of the gabbro are involved in it. The appearance is as if the magnetite had existed previously and been introduced into the mass of fluid gabbro and never generally diffused.

Winchell,[e] describing the different kinds of iron ore found in northeastern Minnesota, says:

> The third variety of iron ore is also magnetite. It is coarse and has a duller luster than the Animikie ore and is not so strongly magnetic. It is found in many places in the gabbro, which sometimes contains so much of it that it seems to be pure mag-

[a] Wadsworth, M. E., Preliminary description of the peridotytes, gabbros, diabases, and andesytes of Minnesota: Bull. 2, Geol. and Nat. Hist. Survey of Minn., 1887, p. 49.

[b] Idem, pp. 63, 64.

[c] P. 304.

[d] P. 305.

[e] Winchell, H. V., Seventeenth Annual Report of the Geological and Natural History Survey of Minnesota, 1888, pp. 80–81.

netite. This ore almost always contains a large amount of titanic acid which ruins it for merchantable ore, with only present methods of reduction. Immense deposits of this titaniferous ore are found, and most of them have been purchased from the United States Government in the hope of being able to conduct remunerative mining operations. When a method for reducing titaniferous ores cheaply is discovered, such iron ore will be valuable; but until then it is worthless.

An analysis of a sample of this ore from section 36–63–10, as reported by Mr. C. F. Sidener, is:

	Per cent.
Silica, SiO_2	11.37
Alumina, Al_2O_3	1.32
Magnetic oxide of iron, Fe_3O_4	53.33
Protoxide of iron, FeO	14.42
Oxide of titanium, TiO_2	16.03
Lime, CaO	.10
Magnesia, MgO	2.73
Phosphorus, P	.01
Sulphur, S	Traces.
	99.31
Metallic iron, Fe	49.40

In this same report he says:[a]

Speaking of the gabbro on Mayhew Lake, this gabbro contains considerable magnetite. Several claim cabins are located on this gabbro ore. The gabbro in Tucker Lake just west of Mayhew Lake is also magnetitic. * * * In the SE. ¼ NW. ¼ sec. 2, 64–3 the gabbro contains parallel bands of magnetite which dip S. 40° and give the rock a decidedly stratified aspect. Some of the beds or bands of magnetite are 4 or 5 inches thick, others only a fraction of an inch.

U. S. Grant, in the report mentioned,[b] refers to another locality on Mayhew Lake. He says:

On the end of the little point which is in the SE. ¼ SE. ¼ sec. 36, the gabbro has changed somewhat, and on the south side of this point it surrounds a large mass of iron ore. This ore seems to be principally magnetite, with a little scattering feldspar, but as it is in the gabbro, it very probably contains a considerable amount of titanium. The exposure of ore was 30 feet wide, and extended for about 300 feet along the shore, rising 15 feet above the water. The contact between the ore and gabbro was found at one place; here the gabbro does not pass into the iron ore by gradually acquiring more magnetite, but there is a sharp and distinct line between the two.

In Bulletin 6 of the Minnesota survey, on the iron ores of Minnesota, four groups of iron ores are recognized, one of which is called the "gabbro titanic-iron group." This is described as follows:[c]

There is certainly no iron ore in Minnesota which is known to exist in larger amounts than this. The explorer in the iron region is continually finding this ore in immense masses, and it has been the cause of many visits by intending purchasers. Fortunately the lithologic characters that surround it are so simple, uniform, and evident that the tyro may discern them. The belt that carries this ore is wide, and it extends

[a] Op. cit., p. 105.

[b] Op. cit., p. 169.

[c] Winchell, N. H., and Winchell, H. V., The iron ores of Minnesota: Bull. 6, Geol. and Nat. Hist. Survey of Minn., 1891, p. 123.

from Duluth to Pigeon Point, constituting, in its culminating topographic points, the summits of the Mesabi Range.

The bulletin[a] states further:

The gabbro titanic-iron group of lithologic characters has been described. It is only necessary to say here that the ore is a black, massive magnetite, generally of coarse, hackly crystalline fracture, and when pure and in large quantities, as it is frequently, it appears in bold and bald knobs, and is dispersed irregularly through the gabbro of the Mesabi Range.

It seems to be only a highly magnetited condition of the structureless gabbro rock itself. This ore was first seen by the officers of the survey at Mayhew Lake, north from Grand Marais. Mayhew Lake is sometimes known as Iron Lake.

This iron is in considerable but unknown quantity at this place, but appears in a similar manner in many places in the vicinity of Mayhew Lake, toward the southwest and west, as well as toward the southeast. This ore may also be seen in the gabbro within the corporate limits of Duluth, where it appears simply as a very highly ferruginous condition of the gabbro, with the other gabbro minerals as its impurities. There is no place within the area of the gabbro sheet where this ore may not exist, as its appearance is entirely fortuitous, but its quality varies from nearly pure magnetite (or menaccanite) to simply magnetitic gabbro.

This report[b] also gives three analyses in addition to those already cited, as follows:

Results of analysis of ore from section 36, 65–3 W. (Iron Lake).

[Analyst, Prof. J. A. Dodge.]

Silica	20.90
Alumina	1.75
Lime	Trace.
Magnesia	.2.63
Titanium binoxide	2.23
Phosphorus	None.
Protoxide of iron	2.01
Magnetic oxide of iron	70.29
	99.81
Metallic iron	52.46

Results of analysis of iron sand from Black Beach, near Beaver Bay.[c]

[Analysts, J. A. Dodge and C. F. Sidener.]

Silica	65.17
Magnetic oxide of iron	30.06
Protoxide of iron	2.23
Dioxide of titanium	2.48
Lime	Trace.
Magnesia	Trace.
Alumina	Trace.
	99.94
Metallic iron	23.50

[a] Op. cit., pp. 135–136.

[b] Op. cit., pp. 141–142.

[c] The statement is made that the sand is derived chiefly from the decomposition of titaniferous gabbro. Bayley states, however, that the rock of Beaver Bay is not a part of the Duluth gabbro, but probably a part of a sheet or sill intrusive in the Keeweenawan, and occurring at a much higher horizon (Jour. of Geol., vol. 2, 1894, p. 819).

Results of analysis of iron sand from the SW. ¼ sec. 33, 61–12, south shore of Birch Lake.

[Analyst, Prof. J. A. Dodge.]

Silica	5.19
Alumina	2.95
Lime	Trace.
Magnesia	.35
Phosphorus	None.
Titanium dioxide	36.77
Metallic iron	41.12

Bayley, in discussing the peripheral phases of the great gabbro mass of northeastern Minnesota, speaks of the titaniferous magnetites. He says:[a]

The specimens in the possession of the United States Geological Survey all contain a large proportion of magnetite, which in some cases runs so high as to constitute 90 per cent of the entire rock. These latter rocks have the bright metallic luster of magnetite and usually a finely granular texture. The luster frequently becomes very brilliant and the specimen takes on the peculiar pinkish tinge of ilmenite. A careful inspection of such specimens as seem to be pure ore reveals the presence in them of tiny yellowish-green grains of olivine, but these are so rare that they produce but little impression on the general aspect of the rock. When the magnetite makes up practically its entire body the rock is much jointed, and, consequently, it easily breaks into small irregular or cuboidal fragments, as Winchell has already noted, and these are not unfrequently natural magnets. Tests of magnetites of this kind from the shores of Little Sasaganaga Lake, NW. ¼ sec. 7, T. 64 N., R. 5 W., prove them to be very rich in titanium, as was surmised from their pinkish tinge.

The ore and its occurrence were described by Elftman in 1898. In a discussion of the minerals of the gabbro, he says:[b]

Magnetite is scattered throughout the rock. Ordinarily it occurs in small grains usually having perfect crystal outlines. Large irregular areas, making up a large part of the rock, are found in some sections. The proportion increases until the entire rock is made up of this mineral. Chemical examinations show that the mineral is sometimes free from titanium, but usually the latter constituent is present in varying amounts up to 20 per cent.

Elftman gives the following description[c] of the occurrence of the ores:

Small masses of titaniferous magnetite are scattered throughout the gabbro. Extensive masses are known only along the northern border and in the eastern part of the gabbro. These deposits of magnetite are especially numerous in the region between Brule Lake and the northern limit of the gabbro. The ore bodies are found in three belts running east and west. The northernmost follows the boundary of the gabbro. The second runs through the southern part of township 64 north, and the southern or third belt runs south of Brule Lake through the central or southern part of township 63 north.

The magnetite deposits are usually associated with granulitic norite, which is always the slightly older of the two. The norite is considerably broken and traversed by

[a] Bayley, W. S., The basic massive rocks of the Lake Superior region: Jour. of Geol., vol. 2, 1894, p. 518.

[b] Elftman, A. H., The geology of the Keweenawan area in northeastern Minnesota: Am. Geol., vol. 22, 1898, p. 140.

[c] Op. cit., pp. 145–146.

veins of magnetite extending from the larger masses around its edge. From the main mass of the magnetite stringers and apophyses run several hundred feet into the massive gabbro. Frequently the magnetite occurs in numerous parallel bands several inches to a foot in width, which alternate with bands of the associated gabbro. As many as 25 of these bands have been noticed within a belt 10 feet wide. There are all proportions of minerals between magnetite and the normal phase of the gabbro. The variations of the ore as seen in a shaft 19 feet deep on section 21, T. 63 N., R. 4 W., are as follows: At the surface the rock was composed of solid coarse magnetite stained green with a coating of malachite. The pure magnetite continued downward for 5 feet, when a few grains of chalcopyrite, pyrite, and plagioclase were found. At 7 feet below the surface the magnetite became fine grained, and a small amount of plagioclase and pyrite continued. Olivine appears as bright, yellow grains. Continuing downward the plagioclase and pyrite disappeared and the lower 10 feet of the shaft were in magnetite, with the olivine increasing in proportion.

The ore has a shining black luster, is brittle, and usually coarse grained. Thin sections of the apparently pure ore generally show the presence of small particles of plagioclase, and sometimes of other minerals. * * *

Chemical analyses show quite a variation in the composition of these ores. An analysis by the writer of an ore from sec. 21, T. 63 N., R. 4 W., gave silica, 6.08; alumina, 3.82; lime, 1.69; titanium oxide, 14.73; magnetite, 71; nickel oxide, 2.65; sulphur, trace. The lime and alumina are accounted for by the presence of a triclinic feldspar.

Titanium is sometimes absent. Cobalt, manganese, and chromium are sometimes found in addition to the constituents given in the above analysis. The metallic iron varies from 49 to 60 per cent. These extensive magnetite deposits at present afford the only ores from the gabbro of probable economic value.

After describing the character of the titaniferous ore deposits in the gabbro, Clements[a] says of them:

The injurious effect of the titanium in rendering the magnetite unmarketable would of course apply to these titaniferous magnetite bodies whatever their size. However, so far as we know, up to the present time no published description has been given of any large continuous masses of the practically pure titaniferous magnetite.

The following recent statement concerning these ores by Van Hise and Leith[b] is in the nature of a summary of existing knowledge, and for that reason is quoted in full:

The great gabbro mass of Lake and Cook Counties, Minn., contains much magnetite, both disseminated and segregated into ore deposits. Complete gradation may be observed between gabbro carrying little magnetite and magnetite carrying little of the ferromagnesian constituents and feldspar. The known deposits are extremely irregular, with gradations between themselves and the gabbro and containing within themselves much gabbro material. They weather very much like the gabbro and might be easily unnoticed on the weathered surface. There has been little exploration for these ores. A few drill holes have been sunk in the region south of Gunflint Lake, some of them revealing depths of ore aggregating several hundred feet. The known deposits seem to be distributed in irregular zones roughly parallel to the north or basal margin of the Duluth gabbro.

[a] Clements, J. M., The Vermilion iron-bearing district of Minnesota: Mon. 45, U. S. Geol. Survey, 1903, p. 421.

[b] Van Hise, C. R., and Leith, C. K., The geology of the Lake Superior region: Mon. 52, U. S. Geol. Survey, 1911, p. 561.

The composition of the ore averaged from 3,560 feet in 14 drill holes is 48.8 per cent iron. The range is from 54 to 20 per cent. The high titanium content renders the ores of doubtful value for the present.

Where the gabbro comes into contact with the iron-bearing Gunflint formation both formations carry magnetite so similar in texture that it is difficult to tell one from the other. However, on analysis the gabbro magnetite is found to be titaniferous, whereas that of the Gunflint formation is not titaniferous. This fact seems to argue against any considerable transfer of material from the gabbro to the iron-bearing formation during its alteration.

The titaniferous magnetites of northeastern Minnesota are direct magmatic segregations in the Duluth gabbro, according to all of the geologists who have studied them, including Irving, Merriam, Bayley, Grant, Winchell, Clements, Van Hise, Leith, and others. The complete gradation from gabbro with a small amount of original magnetite to a magnetite with small amounts of amphibole and other gabbro minerals can be seen in almost any part of the titaniferous magnetite deposits. It is scarcely necessary to repeat the detailed petrologic evidence so fully given by the writers named.

Evidence is given elsewhere for the intrusive character of the Duluth gabbro. It cooled far beneath the surface, where there was not easy escape for its solutions. This fact is taken to explain its retention of its iron oxide. It has been argued under an earlier heading that where basic rocks of similar composition reached the surface large quantities of iron escaped and became available for ordinary sedimentary deposition.

The area underlain by the Duluth gabbro is for the most part an uninhabited region of woods and lakes, where the only means of travel is by canoe. It was impossible and also useless for the purposes of this report to have visited the whole of such a vast area. It was also apparent from the literature that the actual importance of these deposits with respect to size and iron content was open to doubt. It was decided, therefore, to visit the most promising region within the gabbro area. Judging from the published descriptions this seemed to be the area about Iron Lake. Responses to inquiries made of numerous geologists, prospectors, and others who were familiar with the area also pointed to the Iron Lake region as having the most promising showing of ore. It was further learned that considerable drilling had been done there some years ago, and the drill records and detailed maps were very kindly placed at the writer's disposal. Consequently the region about Iron Lake was selected as the one to be visited, and the following paragraphs are an account of the observations made there.

IRON LAKE REGION.

A week was spent in this region with Nicholas Probek, of North Lake, as guide. The familiarity of Mr. Probek with the country made possible the careful examination of a large area. The area examined was the country adjacent to Tucker, Iron, Portage, and Poplar Lakes. It is a narrow belt extending on both sides of the town line between T. 64 N. and 65 N., and from the middle of R. 1

W. to the middle of R. 3 W. For convenience of discussion the region is considered under subheadings corresponding to the lakes above mentioned.

TUCKER LAKE.

Tucker Lake is a narrow lake with its axis lying nearly east-west. It extends from near the western edge of section 4 to near the middle of sec. 1, T. 64 N., R. 3 W. Its eastern end is divided into a northern and southern arm by a narrow peninsula. Its outlet is at about the middle of the north shore.

Along the north and south shores of the lake in the neighborhood of the outlet, and for some distance along the outlet, are almost continuous rock exposures. The prevailing rock is the normal gabbro, in many places with considerable feldspar, so that it has a light color. Magnetite occurs throughout the rock in sparsely disseminated grains. The most prominent femic constituent is the pyroxene, which has usually to a great extent undergone uralitization. Biotite is locally abundant in intimate association with the pyroxene, and seems to be a secondary product. At several places the normal coarse gabbro gives way to a fine-grained rock of brownish-gray color. Thin tablets of feldspar, 1 cm. long and less than 1 mm. thick in parallel arrangement, give this rock at times a schistose appearance. Certain phases of this rock are rather heavy and weather with a black surface suggesting magnetite. A careful examination of this rock, which has been called ore, shows that it contains only a small amount of ore minerals. At only one point, on the south shore, a little west of the outlet, is there anything that could be called ore. The gabbro here contains a small segregation of ore 25 feet long. The greater part of this is very lean, being nothing more than gabbro in which the feldspar has become subordinate and the pyroxene the most prominent constituent. Magnetite is present in considerable amount, and makes up a rock which at first sight appears to be good ore. A closer examination reveals a large quantity of black augite and considerable olivine. A polished section of one of the richest pieces of this ore showed that less than half consists of ore minerals, and this is almost entirely ilmenite.

A half mile north of the outlet at Tucker Lake, in the SE. ¼ sec. 34, T. 65 N., R. 3 W., is an opening in an ore body. The pit has a length of 125 feet along the outcrop, which is here in the form of a ledge. Several hundred tons of ore have been blasted loose and the exposures are very good. The outcrop of ore can be traced about 100 feet east of the pit, beyond which it is covered with drift. To the west of the pit it is entirely covered. This ore body ranges in width from 8 to 10 feet, and passes into gabbro on each side. The ore itself is rather lean, and grades rapidly into normal gabbro, so that a sharp distinc-

tion can usually be made between gabbro and ore. Within the ore body are small inclusions of gabbro with boundaries so distinct against the surrounding ore as to have the appearance of real included fragments of the gabbro country rock and not segregations of the gabbroitic material in the ore. The relations of the contact and these inclusions would suggest that the segregation did not take place where the ore body is now found, but that the ore body was squeezed into its present position, perhaps while the gabbro was still in a molten or viscous state. Between this pit and the shore of Tucker Lake are several exposures showing schlierenlike ore bodies of still smaller size whose relations to the country rock suggest a similar origin. Northwest of the pit is a ledge of gabbro with a small band of lean ore running through it. In the leaner parts of the ore the pyroxenes are the most prominent gangue minerals; in the rich part, consisting almost entirely of the ore minerals, olivine is the most abundant gangue. Analyses of ore from this property are said to have shown about 18 per cent Ti.

Along the north shore of the north arm of Tucker Lake are a number of exposures of gabbro showing considerable iron. The ore occurs in small ferromagnesian-rich areas of the gabbro. These areas are characterized by the feldspar becoming subordinate in amount accompanied by a rapid increase in the amount of pyroxene. At the same time olivine and magnetite become prominent. This rock grades over into ore by magnetite, and, to a less extent, olivine, becoming more abundant at the expense of the pyroxene. The decrease in the amount of pyroxene is, however, more apparent than real as the dark pyroxene is difficult to recognize in the magnetite, whereas the clear yellow olivine stands out strongly against the black magnetite. At one point the ore swarms with fine needles of apatite. Here again polished surfaces show that the ore is not nearly so rich as would appear from hand specimens, and even the richest pieces do not show much more than 50 per cent of ore minerals. Less than 100 yards northeast of the end of this arm of the lake is a cut 8 feet wide, 10 feet deep, and 25 feet long. The best ore seen along this shore occurs there, but there is a rapid alternation of good and bad ore, so that the average of the rock as obtained in mining would be rather lean.

Exposures of highly magnetitiferous gabbro grading into ore occur on the peninsula between the two arms of Tucker Lake and along the north shore of the southern arm to the narrows, draining a small lake that lies to the east and is known as "The Pond." The nature of these outcrops is the same as those on the north shore of the north arm of the lake. The ore bodies, wherever their width is exposed, are narrow, usually not more than 12 feet in width, and their length is not more than 100 to 200 feet, being considerably less in most

cases. Where their limits are not exposed on account of the drift covering, in most cases other outcrops of gabbro near by show that their size is no greater. On account of the dark-colored gangue minerals, hand specimens always appear much richer than is actually the case; and specimens of seemingly good ore are seen on polished surfaces to be rather lean. Apatite is abundant in some of these ores. Analyses at different depths of a diamond-drill core through one of these ore bodies showed the following composition of the ore:

Results of analyses of diamond-drill core from the NE. ¼ NE. ¼ sec. 2, T. 64 N., R. 3 W.

Depth	30 feet.	60 feet.	100 feet.	150 feet.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Iron peroxide	35.00	35.14	35.00	33.71
Iron protoxide	32.92	32.28	32.28	33.41
Alumina oxide	.51	.48	.40	.26
Manganese oxide	2.36	2.89	2.26	2.12
Chromium oxide	.06	.06	.012	.01
Vanadium oxide	Trace.	Trace.	Trace.	Trace.
Magnesia carbonate	1.20	1.25	1.82	1.16
Calcium carbonate	2.16	2.07	2.24	1.14
Phosphorus	.012	.012	.008	None.
Sulphur	.01	.008	None.	None.
Titanic acid	1.25	1.28	.616	.17
Silica	24.00	24.50	26.00	26.00
Metallic iron	50.50	50.00	50.00	50.00

The titanium in these analyses is very low, the average being 0.83 per cent TiO_2. This is not representative of the ore in this vicinity, however, as polished sections show considerable ilmenite. The average iron content in terms of metallic iron is 50.13 per cent. Some of this iron occurs in the femic minerals of the gangue and is not present in the ore minerals. The maximum percentage of magnetite the ore could contain is that corresponding to the percentage of iron peroxide given in the analyses. Assuming all of the peroxide of iron to be present in the form of magnetite, the percentage of iron contained in the magnetite would be 36.75, 36.90, 36.75, and 35.40 per cent, respectively, an average of 37.20 per cent.

At the eastern end of the series of outcrops being discussed, along the shore of the narrows mentioned above, is a ledge of rock 300 feet long, at the eastern end of which is a small pit about 10 feet deep. The western end of this outcrop consists of the normal coarse-grained gabbro; toward the east increasing amounts of magnetite occur, until the richest ore is encountered at the pit. This ore differs from the ore previously described in that the feldspar and ferromagnesian minerals are present in the same ratio as in the normal gabbro. The change in composition of the rock is due wholly to the increase in percentage of magnetite. The feldspar has assumed, however, a distinct tabular habit, giving the rock a coarse diabasic texture. There is also a tendency to parallel arrangement on the part of the feldspar crystals which gives the rock a schistose appearance in

places. Very little good ore occurs here and most of it is too lean to be called ore. A thin section of the ore showed feldspar and femic minerals about equally abundant. The femics are augite and olivine with a little biotite. The biotite seems to be principally secondary, occurring around the edges of the magnetite grains.

This series of outcrops crosses the narrows and continues eastward south of "The Pond" nearly to the line between range 2 and range 3, beyond which the gabbro contains little iron. Outcrops of ore bearing gabbro are numerous, but the extent of each is limited. The prevailing ore is of the feldspathic variety last described, though many outcrops are of better grades. High-grade ore occurs only as narrow seams and stringers in the lean ore. A thin section of the rich ore consisting of about one-third gangue had olivine as the most abundant gangue mineral; it also had considerable augite. Besides these, a little plagioclase and a few flakes of mica were present. In the leaner phases, however, the most striking feature is the tabular crystals of plagioclase. This ore differs decidedly from the Rhode Island ore with its phenocrysts of tabular feldspar. The groundmass is of much coarser grain, so that the feldspar and other constituents are more nearly of the same order of magnitude; and the feldspar crystals are far more numerous, giving the ore a pronounced diabasic texture, which the Rhode Island ore does not possess. The only ore resembling it is that from Iron Mountain, Colo.; but in the latter the feldspar is not so prominent nor the diabasic effect so marked. Plate VIII, *A*, shows a rich piece of this Iron Lake ore with the large tabular crystals of feldspar; Plate VIII, *C*, shows the Iron Mountain ore in which the feldspar tablets are more abundant, but smaller. Analyses of different sections of a diamond-drill core through a body of this ore showed the following composition:

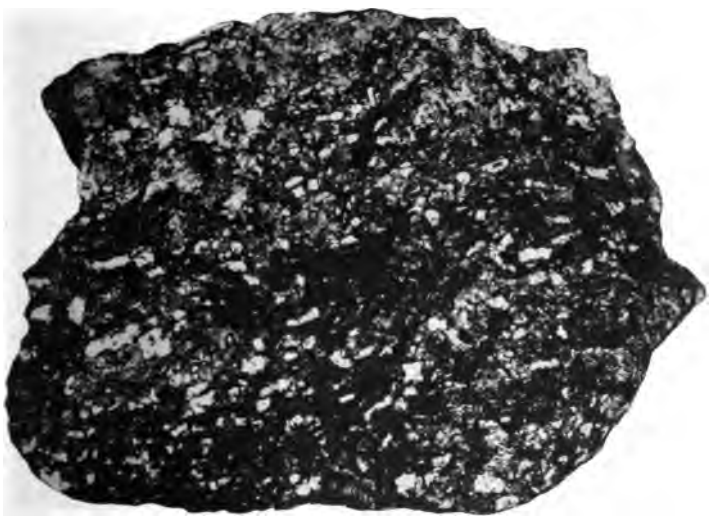
Results of analyses of diamond-drill core from the NE. ¼ NE. ¼ sec. 1, T. 64 N., R. 3 W.

Depth	55 feet.	65 feet.	75 feet.	85 feet.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Iron peroxide	35.61	46.14	35.14	41.43
Iron protoxide	30.00	21.53	32.40	24.05
Alumina oxide	2.70	4.89	4.00	2.20
Manganese oxide	1.75	2.60	3.20	.20
Chromium oxide	1.11	1.01	2.05	1.50
Vanadium oxide	1.006	.906	1.03	2.60
Magnesia carbonate	1.70	1.20	.06	1.54
Calcium carbonate	2.50	.056	.04	.05
Phosphorus	.002	.007	.002	.002
Sulphur	.006	None.	None.	.003
Titanic acid	6.50	None.	8.25	5.75
Silica	17.11	21.65	13.81	20.67
Metallic iron	49.20	49.30	48.40	48.20

The titanium content is rather low, the average TiO_2 content of the four analyses being 5.13 per cent. The average of the metallic iron given in the analyses is 48.78 per cent. It must be remembered,

A. IRON LAKE, MINN., ORE. (× 1½.)

B. POPLAR LAKE, MINN., ORE. (× 2.)



C. IRON MOUNTAIN, COLO., ORE. (NATURAL SIZE.)

however, that some of this iron is present in the ferromagnesian gangue minerals. If we assume all the peroxide of iron to be present in the form of magnetite, the metallic iron contained in the magnetite would be 37.39, 48.45, 36.90, and 43.50 per cent, respectively, an average of 41.56 per cent.

IRON LAKE.

In addition to the ore outcrops just described, at the western end of Iron Lake and along the south shore toward that end there are others farther east along the lake, but only one or two of these is here described.

Extending across a small island southeast of the portage from Loon Lake is a band of ore 200 yards long and 20 to 30 feet wide. Gabbro is exposed on both sides of the ore body. The ore is for the most part lean. Along the north shore, east of the outlet, is a series of outcrops of ore. Some of these show very good ore, but each is of small size, passing within short distances into gabbro. One of the larger of these outcrops was traced for over 200 yards, but its width averaged only 12 to 15 feet. Even the high-grade ore contains inclusions of lean ore and gabbro; and, in other cases, the rock consists merely of gabbro with numerous bands and stringers of ore running through it. A specimen of the high-grade ore from this locality analyzed 53.30 per cent Fe, 24.30 per cent TiO_2.[a]

PORTAGE LAKE.

On Portage Lake little ore was observed. The most prominent outcrop is on the north shore of the north arm of the lake, east of the portage from Iron Lake. At this point there can be traced, for a distance of about 100 yards, a series of small ore outcrops within an area of gabbro. The ore is of the feldspathic variety and contains considerable gangue.

POPLAR LAKE.

Outcrops are abundant along the entire north shore of Poplar Lake, and many exposures of ore were seen. In all cases where the exposures were continuous enough the ore bodies were seen to be of small size. The character of the ore showed considerable variation from place to place. In some exposures the feldspathic ore occurred, in others the ore with the femic minerals. Thin sections showed considerable apatite throughout the ore, and some of the crystals of apatite are large enough to be easily recognized in the hand specimens. In addition to the ore bodies, the gabbro along this shore contains considerable magnetite scattered through it as an accessory constituent.

[a] Analysis computed from analyses of concentrates and tailings made by A. C. Fieldner, of the Bureau of Mines.

COMPOSITION OF THE ORE.

The ore has been described with sufficient detail in the discussion of its occurrences, so that only a brief résumé is here given. Probably the most striking feature of the ore is the great variation it presents both in appearance and composition. The high-grade ore with little gangue tends to be compact and massive, rather than granular. Of the leaner ores, there are the two varieties that can be designated according to the most prominent gangue minerals, the feldspathic and the femic. The femic ores are medium grained and possess no marked characteristics. The feldspathic ores tend to be somewhat coarser grained and are characterized by the coarse diabasic texture produced by the lath-shaped feldspar crystals.

A most interesting feature is the chemical composition of the ore, on account of the unusual behavior of the titanium content. The available results of analyses are tabulated below:

Results of analyses of Minnesota ores.

Constituent.	1	2	3	4	5	6	7	8	9
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Fe	58.48	56.16	55.04	49.83	52.46	23.50	41.12	51.40	53.30
TiO_2	20.15	0–10	4.17–31.67	16.03	2.23	2.48	36.77	14.73	24.30
SiO_2	2.02	2.40	6.20	11.37	20.90	65.17	5.19	6.08	.....
Fe_3O_4	80.78	.....	.....	53.33	70.29	30.06	.....	71.00	.....
Fe_2O_3	.....	.....	.....	.....	.....	.....	.....	.....	.....
FeO	.....	.....	.....	14.42	2.01	2.23	.....	.....	.....
Al_2O_3	2.68	.....	8.85	1.32	1.75	Trace.	2.95	3.82	.....
Cr_2O_3	2.40	.....	.....	.....	.....	.....	.....	.....	.....
Mn_3O_4	.....	.57	.40	.....	.....	.....	.....	.....	.....
CaO	Trace.	.04	.15	.10	Trace.	Trace.	Trace.	1.69	.....
MgO	.....	1.35	2.28	2.73	2.63	Trace.	.35	.....	.....
NiO	.....	.....	.....	.....	.....	.....	.....	2.65	.....
P	.013	.003	.003	.01	.....	.....	None.	.....	.....
S	.....	Trace.	.876	Traces.	None.	.....	.....	Trace.	.....
	.....	.....	.....	99.31	99.81	99.94	.....	.....	.....

Constituent.	10	11	12	13	14	15	16	17
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Iron peroxide	35.00	35.14	35.00	33.71	35.61	46.14	35.14	41.43
Iron protoxide	32.92	32.28	32.28	33.41	30.00	21.53	32.40	24.05
Alumina oxide	.51	.48	.40	.26	2.70	4.89	4.00	2.20
Manganese oxide	2.86	2.89	2.26	2.12	1.75	2.60	3.20	.20
Chromium oxide	.06	.08	.012	.01	1.11	1.01	2.05	1.50
Vanadium oxide	Trace.	Trace.	Trace.	Trace.	1.006	.906	1.03	2.60
Magnesia carbonate	1.20	1.25	1.82	1.16	1.70	1.20	.06	1.54
Calcium carbonate	2.16	2.07	2.24	1.14	2.50	.056	.04	.05
Phosphorus	.012	.012	.008	None.	.002	.007	.002	.002
Sulphur	.01	.008	None.	None.	.006	None.	None.	.003
Titanic acid	1.25	1.28	.616	.17	6.50	None.	8.25	5.75
Silica	24.00	24.50	26.00	26.00	17.11	21.65	13.81	20.67
Metallic iron	50.50	50.00	50.00	50.00	49.20	49.30	48.40	48.20

Of these analyses the average Fe content is 49.23 per cent and the average TiO_2 content 10.96 per cent. The average titanium content is somewhat lower than the average for titaniferous magnetites. It would be fairer, however, to take an average of analyses 10, 11, 12,

and 13 and count that as one analysis, and the same with 14, 15, 16, and 17, as they represent only different parts of a drill core. If the analyses are averaged on this basis, there is practically no change in the iron, but the average titanium content becomes 13.23 per cent TiO_2, a percentage that is more nearly the normal. The unusual feature of this ore is the wide range shown by the titanium. In general, the titanium present in a given deposit is fairly constant. In the case of these ores, the percentage of titanic acid ranges from zero to nearly 37 per cent. Phosphorus is present in the form of apatite in considerable amount locally, but averages low. Another point of note is the absence of spinel in the thin and the polished sections examined.

METALLOGRAPHIC DESCRIPTION OF THE ORES.

Polished sections of the ores show considerable uniformity as regards the intergrowth of magnetite and ilmenite. Otherwise, as one would expect from the description of the ores, there is considerable variation. As is generally the case, polished sections of the femic ores show more gangue than is apparent in the hand specimens; and there is every gradation from pure ore wholly lacking in the essential constituent in the gabbro to rock that can be called nothing more than magnetite-bearing gabbro.

The ratio of magnetite to ilmenite also varies from ore in which ilmenite is almost lacking to ore in which magnetite is almost lacking.

The characteristic of most of the polished sections is the minute association of the titanium in the magnetite with the magnetite. Very little dull-black magnetite occurs. Most of it shows a dull-gray mottled effect to a bright iridescent effect. These effects pass over, however, by gradual transitions, to minute networks, and these into coarser networks. Also by increasing the magnification sufficiently seemingly continuous iridescent surfaces can be resolved into a close minute network of ilmenite against a black background of magnetite. Plate VII, *B*, shows an area of gangue and one of magnetite with ilmenite intergrowths. Adjacent to the gangue area, the ilmenite is so closely intergrown as to form a nearly continuous bright surface. This passes over into a delicate network. Plate IX, *A*, shows a well-developed minute delicate network, which with less magnification gives the magnetite grain a semilustrous appearance.

Though these minute intergrowths are the prevailing ones, other forms also occur. Some of the sections show beautifully developed network of the normal coarseness. Others show very coarse open network passing over into areas containing only isolated thick lamellæ. Plate VIII, *B*, shows an ore of an unusually coarse network. The thin border of ilmenite about the outline of magnetite grains, as illustrated in Plate IV, *A*, also occurs.

POSSIBILITIES OF UTILIZATION.

Deposits of titaniferous magnetite are very numerous in northeastern Minnesota, but as a general proposition it may be stated that the high-grade deposits are too small and the large deposits too lean to work.

To utilize the large deposits magnetic concentration would be necessary, but the nature of the ore is such as not to promise satisfactory results. From deposits in which nearly all of the ore consists of ilmenite, considerable titanium could be removed, but the concentrates would form only a small part of the original material. The ore of other deposits in which the magnetite forms a considerable percentage of the ore is usually swarming with the minute inclusions of ilmenite just described. Any considerable diminution of the titanium content in such ore is out of the question. Below are presented results of a separation made of a specimen showing only few grains of ilmenite and bearing magnetite that exhibited the dull, mottled, and bright iridescent surfaces characteristic of the ore from the Iron Lake region.

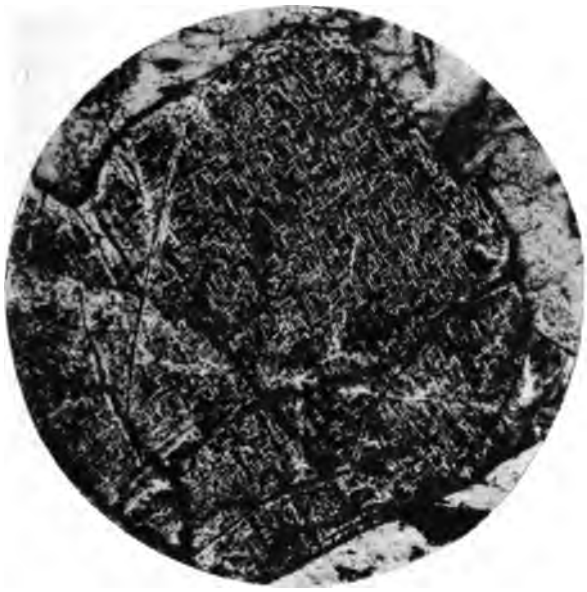
Results of magnetic concentration of Iron Lake ore.

	Unscreened ore.	Ore through 50-mesh, over 100-mesh.[a]		Ore through screen finer than 100-mesh.[b]
		Magnetic.	Non-magnetic.	Magnetic.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Quantity	.....	93.3	6.7	92
Fe	53.30	54.93	30.59	54.13
TiO_2	24.30	23.15	40.37	23.58

[a] 38.5 per cent. [b] 61.5 per cent.

These results confirm the intimate relation of magnetite and ilmenite brought out by the metallographic study.

In addition to the disadvantages of these deposits already mentioned, their inaccessible location must also be taken into consideration.



A. TUCKER LAKE, MINN., ORE. (X 60.)

B. IRON MOUNTAIN, WYO., ORE. (X 2.)

WYOMING.

IRON MOUNTAIN.

Iron Mountain is situated in the southeastern part of Wyoming, 20 to 25 miles northeast of Laramie and 9 miles west of Iron Mountain, on the Colorado & Southern Railroad. A second similar, but smaller, deposit occurs on the Shanton ranch, about 5 miles to the southwest. The two localities can be reached by taking the Sibylee stage from Laramie and stopping at Wayside. The Shanton occurrence lies about 3 miles from Wayside, and Iron Mountain about 8 or 9 miles, by trails which are easy to follow. Iron Mountain can be as readily reached from Iron Mountain station.

The occurrence of magnetic iron ore at Iron Mountain has been known for a great many years, and numerous accounts, more or less complete, have been given of the deposit.

Stansbury [a] as early as 1853 mentioned the occurrence of immense numbers of rounded black nodules of magnetic iron ore which seemed of unusual richness in the bed of Chugwater Creek and on the side of the adjacent hills.

In 1870, Hayden [b] called attention to the deposits in the following words:

> Near the sources of the Chugwater are some very rich iron mines, which may prove of great value to the country in the future. The fact of their existence has been known for some years, but no definite knowledge of them has been given to the world. * * *. In the winter of 1859–60, while attached to the exploring expedition of Gen. W. F. Raynolds, I made a trip to the sources of the Chugwater, and found great numbers of these worn masses of iron ore; but not until a comparatively recent period were they traced to their source in the mountains. During the construction of the Union Pacific Railroad some of the engineers visited the mines and spoke of their future value. In the summer of 1868 I had an opportunity of examining this region in company with Dr. Latham and Judge Whitehead, of Wyoming, and found the mines much richer and more extensive than had previously been supposed. We commenced our examination in the valleys of the smaller branches of the "Chug" as they emerge from the mountains, and found that the stray masses of iron ore were confined to one of them. Following the branch up into the range we soon came to the ore beds themselves, which we found to be interstratified among the red feldspathic granites which compose the nucleus of the range. The ore beds incline in the same direction with the granites, and have the same joints and cleavage, and the examples of slickensides are numerous. They are not continuous, and are confined to a restricted area, yet Mr. Whitehead traced one of the beds a distance of over 1½ miles. The ore is located much like that in the Lake Superior region, and is probably of the age of the Laurentian rocks of Canada. The quantity of ore in this locality appears to be unlimited. Thousands of tons have been washed down into the valley of the "Chug" and distributed among the superficial drift. As we leave the ore beds themselves these stray masses are larger and more angular, and as we pass down the "Chug" they dwindle to minute pebbles

[a] Stansbury, Howard, Exploration and survey of the valley of the Great Salt Lake of Utah, Washington, 1853, p. 266.

[b] Hayden, F. V., Preliminary report of the United States geological survey of Wyoming and portions of contiguous territory, 1870, p. 14.

and disappear. Mr. J. P. Carson, of New York, an assistant in the survey of 1868, made the following analysis of this ore, at the school of mines, Columbia College:

Sesquioxide of iron	45.03
Protoxide	17.96
Silica	.76
Titanic acid	23.49
Alumina	3.98
Sesquioxide of chromium	2.45
Sesquioxide of manganese	1.53
Lime	1.11
Oxide of zinc	.47
Magnesia	1.56
Sulphur	1.44
Phosphorus	Trace.
	99.78
Fe	45.49

King [a] briefly refers to and Hague [b] describes the occurrence of a gabbro consisting essentially of labradorite and diallage which protrudes the granitoid rocks in the form of low irregular-shaped domes and knolls east of Iron Mountain and Chugwater and Horse Creeks. A microscopic description of this rock is given by Zirkel.[c] Hague does not seem to have realized, however, that this rock forms the wall rock of the ores, as he describes the deposit as follows: [d]

Iron Mountain, to which reference has already been made, is a mass of titaniferous iron, or ilmenite, and is situated just north of the Chugwater Creek, about 1½ miles above the point where the stream leaves the hills. The mountain rises about 600 feet above the stream bed, is irregular in form, but has a somewhat oval-shaped outline. It occurs intercalated in the granite, standing nearly vertical, with the walls in places sharply defined; this is the case in the canyon, where the dark iron body resembles a broad dike, which rises to the top of the canyon wall. Frequently large masses of granite are nearly incased in the iron, and again the iron body puts out into the surrounding granite. The main deposit of iron is nearly a quarter of a mile in length, with a strike a little to the west of north and east of south. To the north the main deposit terminates somewhat abruptly; but southward it crosses the canyon, and may be traced cropping out through the granitoid rocks with the same general strike for nearly 2 miles, in the direction of Pebble Creek. These outcrops vary much in size, mostly mere narrow seams and small irregular patches of iron, which disappear in the surrounding granite. Still farther to the south, just above Horse Creek, considerable deposits again make their appearance, but much smaller than Iron Mountain, and, like the latter, have been held for valuable mineral bodies; they are probably only a continuation of the larger one. The ilmenite occurs chiefly as a compact massive deposit, iron-black in color, with a submetallic luster. It is frequently found, however, with a coarse granular structure. It is accompanied by small amounts of magnetite and hematite, which decompose and give portions of the mass a brownish-red appearance. Prof. O. D. Allen, of Yale College, examined specimens of the Iron Mountain ore, and found it to contain a mixture of ferrous and ferric oxide, which gave 50.83 per cent of metallic iron, combined with 23.32 per cent of titanic acid.

[a] King, Clarence, United States geological exploration of the 40th parallel, vol. 1, 1878, p. 27.
[b] King, Clarence, op. cit., vol. 2, 1877, p. 13.
[c] King, Clarence, op. cit., vol. 6, 1876, pp. 107–108.
[d] King, Clarence, op. cit., vol. 2, 1877, pp. 14–16.

Other samples of the ore, analyzed by Prof. Richards, of the Institute of Technology in Boston, gave the following results:

Ferrous oxide	24.55
Ferric oxide	48.97
Titanic acid	23.18
Sulphur	.03
Residue insoluble in acid	2.15
	98.88
Metallic iron	53.33

Samples of the coarse granular variety, collected from the deposits south of Iron Mountain, yielded Mr. R. W. Woodward the following:

Metallic iron	34.29
Titanic acid	49.47

All the samples examined gave a very high, although a varying, amount of titanic acid.

This high percentage of so refractory a substance as titanic acid renders the vast deposits of iron of but little use for practical purposes in iron smelting, which is to be regretted, as the beds in the Laramie Hills could be easily mined, and are so well located in reference to a market and the known resources of iron in Wyoming are so limited.

An account in the Tenth Census is quoted direct from Hayden's description.[a]

The next mention of the deposit is made by Knight,[b] who gives the following results of analysis:

Results of analysis of Iron Mountain ore.

	Per cent.
SiO_2	1.21
TiO_2	22.43
Fe_2O_3	47.21
FeO	25.80
S	1.14
	97.79
Fe	51.72

Kemp[c] reviews the information available in 1899. Commenting on Hague's description, he says that the relations strongly suggest an intruded origin, and that "granite is an unusual wall rock for titaniferous magnetites, but gabbro, it is interesting to note, occurs in the immediate vicinity." He also mentions a rock obtained from Prof. Knight labeled the wall rock of the titaniferous ores which in thin section is a typical gabbro consisting mostly of labradorite[d] and says: "Olivine gabbro would be a much more natural

[a] Tenth Census, 1886, vol. 15, p. 485.

[b] Knight, W. C., Bull. 14, Univ. of Wyoming Exp. Station, 1893, p. 177.

[c] Kemp, J. F., A brief review of the titaniferous magnetites: Sch. of Mines Quart., 1899, pp. 352-355.

[d] Hill, B. F., Notes on a set of rocks from Wyoming collected by Prof. Wilbur C. Knight, of the University of Wyoming: Sch. of Mines Quart., vol. 20, 1899, p. 364.

wall rock than granite, and if it is the wall rock in some of the exposures, it would establish the kinship of the ore with the gabbro, and kinship that anyone familiar with the ores of this type would at once believe and assume." Kemp further calls attention to the notable percentage of zinc in the ore as determined in Carson's analysis cited by Hayden, and compares it with the Rhode Island ore in this respect. Lindgren[a] also calls attention to the fact that the presence of zinc is unusual in titanomagnetites.

Lindgren's paper contains the first definite statement of the character of the deposit, and the next four paragraphs are quoted from it.

Most of the deposits of titanic iron ore form irregular masses or fairly sharply outlined streaks in gabbro, or still more commonly in anorthosite (labradorfels). Distinct dikes, undoubtedly indicating separate igneous injection of molten magma of titanic iron ore, have, however, also been described by Kemp from the Calamity Brook district in the Adirondacks, and by Vogt from near Ekersund, Norway, and the locality in Wyoming is chiefly interesting as belonging to this type.

The rocks prevailing here are chiefly Paleozoic limestones and sandstones, in rolling folds, and these continue for 6 or 7 miles up Chugwater Creek. The dips here become steeper and the underlying pre-Cambrian rocks appear. As far as my observations went they consist exclusively of a labradorite rock of coarse grain which forms rough gray outcrops. The rock can scarcely be called a gabbro, for the pyroxene grains are very sparingly distributed. It contains very little magnetite or ilmenite. Going up 1 mile farther, the chief deposit is encountered; it crosses the canyon of the Chugwater as a solid dike 100 to 200 feet wide, and can be seen extending up several hundred feet in elevation on both sides of the creek. The mass is said to be traceable for half a mile north and south of Chugwater Creek. The amount of iron ore in sight is most remarkable. The contacts are not exposed to the best possible advantage, but have the appearance of being sharp and well defined. The black titanic iron ore seems entirely pure and free from accompanying minerals; at least a search along the base of the outcrop revealed no other constituents of the mass.

About 400 feet below the main deposit there is exposed on the southern wall of the canyon a smaller dike only about 10 feet wide. The contacts are well exposed and show a medium-grained gray labradorite rock abutting sharply against the dike of titanic iron ore. The dike does not continue on the north side of the canyon, the bottom of which is filled with considerable débris. The greater part of the width of the dike is composed of massive titanomagnetite; but adjoining the western contact the iron ore for a width of 1 or 2 feet contains large, imperfect crystals of olivine embedded in a cementing mass of the black mineral. This association of olivine and titanomagnetite is somewhat unusual; Prof. Vogt, in fact, declares that it is not known to occur (Z. f. Prakt. Geol., 1900, p. 292) in the differentiated ores. The black mineral immediately adjoining the olivines contained, upon qualitative test, a large amount of titanium.

Evidently the points where Mr. Hague saw the deposits were not the same as the locality here described, for the dike, as noted, extends over a considerable distance. The only granitic rock observed at the place described was a small, dikelike mass of fine-grained biotite granite on the north side of the canyon, nearly opposite the smaller dike of iron ore.

On November 21, 1904, Kemp presented an abstract of a paper on "The Titaniferous Magnetite in Wyoming"[b] before the New York

[a] Lindgren, W., A deposit of titanic iron ore from Wyoming: Science, vol. 16, 1902, pp. 984-985.
[b] Amer. Geol., vol. 25, 1905, p. 64.

Academy of Sciences, which appeared the following winter in the Zeitschrift für praktische Geologie.[a] This paper agrees substantially with the views expressed by Lindgren and describes the occurrences in greater detail. The following abstract gives the essential features of the paper:

The geology of the region is roughly indicated in figure 1. The anorthosite belt extends in a northeasterly direction for a distance of nearly 20 miles without showing any appreciable variations. A plagioclase of the composition bytownite-labradorite makes up almost the entire rock. As accessory constituents occur monoclinic pyroxene, a little hornblende, some chlorite, a little calcite, and a few grains of titaniferous magnetite. The anorthosite is cut by granite dikes of granophyric texture, which seem to be somewhat more resistant to erosion and stand out at a height of about 1 meter above the anorthosite. Two such dikes cut the ore body on Iron Mountain. One of these is a biotite granite consisting of microcline, quartz, biotite, and a little plagioclase. Titaniferous magnetite occurs at two localities. One of these is at the Shanton ranch, about 6 miles southwest of Iron Mountain, where the ore outcrops in the form of a black wall above its surroundings. The largest ore body has a length of 500 meters and a width varying from 25 meters to zero, with an average width of 6 meters. The analysis made by Woodward and cited by Hague may be of a sample from this occurrence.

A careful examination of the outcrop failed to reveal any mineral other than the magnetite, which shows a smooth polished surface as a result of the sand and dust blown over it by the violent winds. Thin sections of the ore reveal the presence of many inclusions of green spinel, which suggests the magnetite spinelite of Routivara[b] and the mines at Peekskill on the Hudson.[c]

The most important outcrop of the ore is that at Iron Mountain, which is cut into two parts by Chugwater Creek, the larger part lying to the north of the creek and having a northwesterly strike. The ore projects as a black dam across the anorthosite, and forms the ridge of the hill for a distance of about 2 miles. The contact of ore and country rock is everywhere smooth and even slickensided and free of all transitions. The summit of the hill rises about 1,000 feet above the valleys. On the slope of the hill are two subordinate ore dikes. With the exception of the locality next described, the ore is free from all gangue minerals.

On the south side of Chugwater Creek, a short distance east of the main ore body, is a smaller dike about 15 feet wide which carries con-

[a] Die Lagerstätten titanhaltigen Eisenerzes im Laramie Range, Wyoming, Vereinigten Staaten: February, 1905, pp. 71–80.

[b] Petersson, W., Geol. Fören Förhandl., vol. 15, 1893, p. 45.

[c] Williams, G. H., Am. Jour. Sci., March, 1887, p. 197.

siderable olivine. The olivine occurs irregularly, but is more abundant near the periphery of the dike. No change in the character of the anorthosite is noticeable at the contact, which is well exposed here.

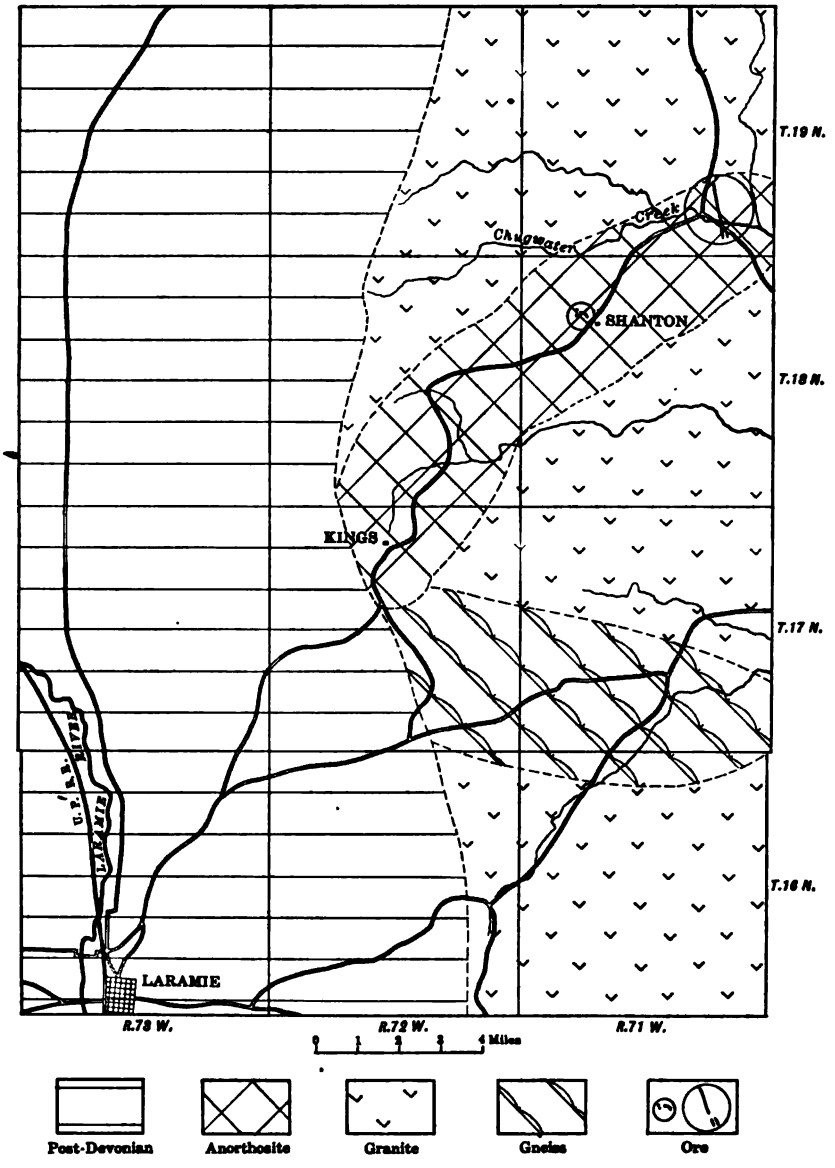


FIGURE 1.—Geological sketch of the region between Laramie and Chugwater Creek, Wyo. (after Kemp).

Though the titaniferous magnetites are usually looked upon as basic segregations, the manner of occurrence of this ore body establishes beyond doubt its intrusive character. It must be looked upon as a

dike. The occurrence of such an ore body is not as surprising as might seem at first thought. Titanium plays the same rôle in eruptive rocks as silica, and these ores have an average of 23 per cent TiO_2. In other words, they are an iron titanate instead of ferromagnesian silicates.

The chemical composition of the anorthosites is likewise significant. They contain little besides silica, alumina, lime, and soda. In a normal rock we would also expect iron and magnesium and a small percentage of TiO_2. If an original simple magma gave rise through differentiation to these two contrasting formations, which in their chemical relation are so abnormal, it was nothing less than exceptional. Therefore, it is not surprising that a titaniferous magnetite is found where the chief product is a ferromagnesian free anorthosite. Also a small iron and titanium content distributed through a large mass can through concentration bring about the formation of a very considerable ore body.

The deposit was also visited and described in 1906 by Ball.[a] He gives some data not contained in the descriptions already discussed, and these are repeated here.

Iron Mountain is described as a rugged ridge 300 to 600 feet wide and 1¼ miles long, its exact location being given by Ball as in sections 22, 23, 26, and 27, T. 19 N., R. 71 W. He says further:

Besides this main body of the magnetic iron, minor masses occur in the pre-Cambrian rocks in a belt which is reported to extend from Horse Creek to Sibylee Creek, a distance of 20 miles. This belt, which courses a little east of north and west of south, is in places 5 miles wide. * * *

The greater portion of the main deposit passed into the hands of the Union Pacific Railroad as part of the land granted to it in 1862. In 1872 a wagon road was built to the deposit, prospectors rushed in, and the whole countryside was staked. In the following year a post office, Iron Mountain, was established at the base of the Iron Ridge, but was abandoned in 1874. Eight or ten years ago the Colorado Fuel & Iron Co. employed 15 teams for several months in hauling ore from the mountains to the railroad, where it was shipped to their smelters at Pueblo. The work was suddenly abandoned, however, although the same company is reported to have made a small shipment four years ago. In 1905 and 1906 the main ore body was visited by a number of surveying corps, and the Colorado Land & Iron Co. is said to have located claims between Chugwater and Sibylee Creek.

The pre-Cambrian complex near the large dike of Iron Mountain consists of three granular igneous rocks—an anorthosite, the iron ore, and a granite. The anorthosite is the oldest of these and is cut by dikes and lenticular masses of iron ore and granite. The relative ages of the iron ore and granite was not certainly determined, since exposures are poor where the two rocks are close together. All of the available evidence, however, indicates that the iron ore is older than the granite.

The mass of iron ore is an igneous dike 1¼ miles long and 40 to 300 feet wide, the greatest observed width being at the point where Chugwater Creek cuts through the mass. The dike trends east of north; most of it lies north of the creek. It widens and contracts rather abruptly throughout its course. Toward the north it gradually narrows and finally disappears, while 300 feet south of Chugwater Creek it narrows slightly and then abruptly ends. At several places it is almost cut in two by wedge-like masses of granite, but throughout practically its whole length it is bordered by

[a] Ball, S. H., Titaniferous iron ore of Iron Mountain, Wyo. Contributions to Economic Geology: Bull. 315, U. S. Geol. Survey, 1907, pp. 206-212.

anorthosite. The contact between the anorthosite and the ore, where exposed, is sharp, neither rock having undergone important gradational changes. A second dike of iron ore is exposed 300 feet downstream (east) on the south side of Chugwater Creek; this dike varies in width from 6 to 20 feet and is clearly intrusive in the anorthosite, with which it has sharp contacts. About one-eighth of a mile southeast of the south end of the main mass is a third dike of iron ore in anorthosite. The trend of this dike, which is 300 feet long and from 10 to 30 feet wide, is approximately parallel to the main mass. Several small magnetite dikes that are from 10 to 50 feet long, and have maximum widths of 3 feet, lie east of this mass in parallel alignment.

The iron ore is a black, granular, holocrystalline igneous rock, with constituent grains varying from one-eighth to one-half inch in diameter. It has a metallic or submetallic luster. Changes in granularity occur in irregular masses or along well-defined parallel planes. In consequence of this distribution the rock has at some places an original gneissic structure. The greater portion of the iron is free from mechanical impurities, but biotite, olivine, and feldspar are sporadically distributed throughout its mass. Olivine is particularly abundant in portions of the small dike 300 feet east of the main mass. The iron ore is cut by rather closely spaced joints and by slickensided fracture planes. In consequence, the outcrop is angular, and its surface is littered with square blocks broken from the ore in place.

As seen under the microscope the ore in some specimens consists principally of titaniferous iron with a little spinel. Other thin sections show considerable olivine, while biotite and labradorite are present in many specimens and in a single section a crystal of what appears to be brown hornblende was noticed.

The writer considers the iron ore and the anorthosite differentiation products of a common magma, the iron ore having been intruded in the anorthosite after that rock had completely solidified.

The above-quoted abstracts of articles by Lindgren, Kemp, and Ball show that their views as to the nature of the deposit are essentially in accord. There are a few inconsistencies in details of description. To serve as a summary of this review, a brief description of the deposits follows, based principally on the writer's own observations.

IRON MOUNTAIN DEPOSITS.

Iron Mountain forms part of the eastern edge of a pre-Cambrian area cut off on the east by Paleozoic limestones and sandstones. The prevailing rocks of the pre-Cambrian area are gneisses and granites, and these have been intruded by a large mass of anorthosite, which has itself been intruded by granitic dikes. Iron Mountain is a ridge a little over a mile long with a direction a little west of north, and an altitude of about 1,000 feet. It is made up principally of anorthosite into which has been intruded a large mass of magnetic ore, which, on account of its greater resistance to erosion, forms the summit of the ridge. The width of the ore body varies considerably, ranging from 50 to 200 feet. The ridge of the mountain is not a straight line but consists of a series of knolls which correspond to the wide parts of the ore body, such places offering greater resistance than where the ore body is narrow. The southern part of the hill is cut through by Chugwater Creek, forming a steep gorge, in

the sides of which the ore body is well exposed. About 500 feet before reaching the creek the dike takes a sudden turn of 200 yards to the west and then resumes its normal strike. Plate X, *A*, shows the part of the dike south of the offset as seen from the south side of Chugwater Creek. The hill in the right background is the main ridge of Iron Mountain. The plate shows clearly the manner in which the ore projects above the adjacent rock. About 400 feet south of the creek the ore body suddenly terminates.

In thin section the anorthosite is a medium to coarse grained rock consisting almost entirely of plagioclase feldspar. As accessory constituents it contains small amounts of magnetite, biotite, and a monoclinic pyroxene. All of these minerals have been somewhat altered, giving rise to the usual secondary minerals. A little south of the center of Iron Mountain are two intrusions of a biotite granite which seemingly cut the ore body. This rock in thin section is seen to consist of microcline and quartz, with considerable orthoclase and biotite.

At the same part of the hill where the granitic intrusions occur, are three smaller ore bodies on the west slope. One of these is only 200 feet below the summit, and the others nearer the foot of the hill.

On the south side of Chugwater Creek, 300 feet east of the main dike, is a smaller dike which is characterized by an unusually large amount of olivine and plagioclase, and shows exceptionally well the contact of ore and anorthosite. Plate X, *B*, is a view of the contact here exposed. The rock on the right is normal anorthosite, which abuts directly against the iron ore without showing any change whatever. That the ore body is not a segregation in situ but an intruded body is evident from this exposure. On the left side of the plate a part of the dike is shown. Against the contact is a band of ore about 1 foot wide that contains such a large quantity of olivine that the olivine in places makes up half of the bulk. Next to this is an extremely crumbly layer about 2 feet wide. This contains, in addition to olivine, considerable plagioclase of the same gray color characteristic of the anorthosite. This part is a granular aggregate of magnetite, olivine, and feldspar, though the magnetite and olivine tend to be more intimately associated with each other than either is with the feldspar. Following this is another band consisting of magnetite and olivine, and then solid ore containing no more gangue than the main ore body. A thin section of the olivine-rich ore from this locality consisted principally of magnetite and olivine. In the magnetite were a few dark-green spinels and two small flakes of biotite. The biotite occurs intimately associated with spinel, the spinel almost surrounding and thus separating them from the magnetite. In general the ore of the main ore body is a

medium to coarse grained magnetite containing little visible gangue. Locally, olivine is rather abundant, especially on the northern end of the mountain. Biotite is also widely distributed, but not in great amounts. On polished surfaces considerable spinel is seen disseminated through the magnetite, and in thin sections this is still more prominent. The crystals attain a diameter of as much as 1 mm. In one slide containing more olivine than the normal amount several small crystals of hornblende occurred in the olivine. A white crust of calcium carbonate frequently coats the ore, especially in crevices. In the northwest corner of the large outcrop on the highest point of Iron Mountain is a layer from 1 to 2 inches thick which cuts across the ore body at an angle parallel to the hill slope. This is an intimate mixture of fragments of ore and bunches of chlorite in a matrix of the white calcium carbonate.

The following results of analyses of the ore from Iron Mountain and from Shanton ranch are available:

Results of analyses of Iron Mountain ore.

Constituent.	1	2	3	4	5	6	7	8	9	10
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
TiO_2	23.49	23.32	23.18	49.47	22.43	21.75	19.47	21.85	21.08	19.98
SiO_2	.76	.....	a 2.15	.....	1.21	.....	.....	.....	.....	.....
Fe_2O_3	45.03	.....	48.97	.....	47.21	.....	.....	.....	.....	.....
FeO	17.96	.....	24.55	.....	25.80	.....	.....	.....	.....	.....
Al_2O_3	3.98	.....	.....	.....	.....	.....	.....	.....	.....	.....
Cr_2O_3	2.45	.....	.....	.....	.....	.....	.....	.....	.....	.....
Mn_3O_4	1.53	.....	.....	.....	.....	.....	.....	.....	.....	.....
CaO	1.11	.....	.....	.....	.....	.....	.....	.....	.....	.....
MgO	1.56	.....	.....	.....	.....	.....	.....	.....	.....	.....
ZnO	.47	.....	.....	.....	.....	.....	.....	.....	.....	.....
P	Trace.	.....	.....	.....	.....	.....	.....	.....	.....	.....
S	1.44	.....	.03	.....	1.14	.....	.....	.....	.....	.....
	99.78	.....	98.88	.....	97.79	.....	.....	.....	.....	.....
Fe	45.49	50.83	53.33	34.29	51.72	52.31	54.48	51.93	51.50	52.98

a Insoluble.

Analyses 1, 2, 3, 4, and 5 have been cited in the foregoing pages, 1 being from Hayden's report, 2, 3, and 4 from Hague's, and 5 from Bulletin 14 of the Wyoming experiment station. Kemp suggests that 4 may be from the Shanton ranch occurrence. Analyses 6, 7, 8, 9, and 10 were computed from analyses of concentrates and tailings made by A. C. Fieldner, of the Bureau of Mines. They are analyses of specimens of ore that were studied metallographically. The first three were from Iron Mountain and the last two from Shanton ranch.

With the exception of analysis 4, the analyses show considerable uniformity in the composition of the ore. Analysis 4 contains iron and titanium in almost the exact ratio in which they are present in ilmenite, but the metallographic study has failed to reveal any ores of that composition at either Iron Mountain or Shanton ranch. It

A. ORE ON IRON MOUNTAIN, WYO.

B. CONTACT OF ORE AND ANORTHOSITE, IRON MOUNTAIN, WYO.

is more likely, therefore, that the sample was obtained from some other small deposit running unusually high in titanium, and it will be left out of consideration. The average composition of the ores from both occurrences is, then, 51.38 per cent Fe and 21.84 per cent TiO_2. The titanium content is somewhat higher than the average for titaniferous magnetites. The average composition is equivalent to an ore composed of 49.84 per cent magnetite and 41.5 per cent ilmenite. With the exception of analysis 1, all the analyses are so incomplete that little further comment can be made on them. The high percentages of sulphur in analyses 1 and 5 are noteworthy, as is the unusual percentage of ZnO in 1, and the presence of several per cent of Cr_2O_3.

The contact between the ore and the wall rock is not usually exposed, but wherever the relations can be seen it is sharply defined. There is no transition from ore to wall rock. The anorthosite in contact with the ore shows the same features that characterize the anorthosite at a distance from the contact. The ore also is the same at the contact with the anorthosite as farther within the ore body. That the deposit is not a magmatic segregation in situ is evident, and the relations are such as to establish beyond doubt the intrusive character of the ore. The relations existing between anorthosite and deposits of titaniferous iron ores in other regions as in Canada, the Adirondacks, Minnesota, and elsewhere, make it most probable that the anorthosite and iron ore came from the same parent magma. Deep-seated magmatic segregation took place. The anorthosite was intruded, and subsequently the iron ore body was intruded into it in the same manner in which the igneous dikes were intruded. The Iron Mountain deposit is therefore the analogue in basic igneous rocks of the large deposits in acidic igneous rocks, of which Kiruna, in Sweden, is the most striking example.

SHANTON RANCH DEPOSITS.

At Shanton ranch are several smaller dikes of similar character to the Iron Mountain deposit. They are situated in sec. 8 or 9, T. 18 N., R. 71 W. The geological conditions here are similar to those at Iron Mountain. The deposit lies within the anorthosite belt, which here is also cut by granitic dikes, one of which is exposed within 50 yards of the ore. The deposits occur in the midst of a comparatively level stretch of land above which they rise as small hills, the summits of which are formed by the iron ore. The relative position of the ore body is shown in figure 2. Outcrops A, B, and C are probably parts of the same ore body, though the exposures are not continuous. Outcrops D and E are smaller isolated occurrences. The width of these varies from 20 to 80 feet, the maximum width

being attained by outcrop C. The relations of the ore to wall rock are the same as at Iron Mountain, and the same explanation of their origin holds.

Results of analyses of the ores have been given in the table on page 120. From these it is seen that they are essentially of the same composition as the Iron Mountain ores. Macroscopically the ores from the two localities are also identical, except that no olivine or feldspar was seen at this locality and only a few flakes of biotite. In almost all specimens any evidence of gangue minerals is lacking. On polished surfaces, however, the presence of spinel is revealed.

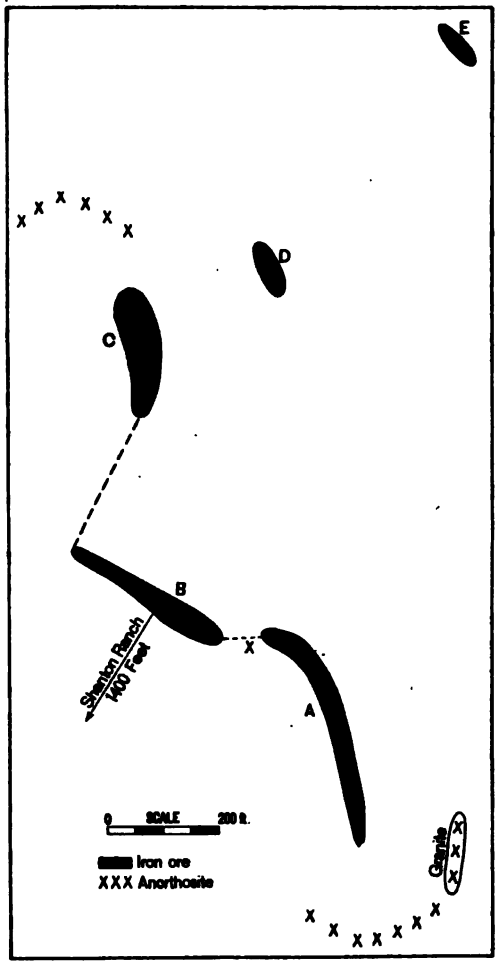
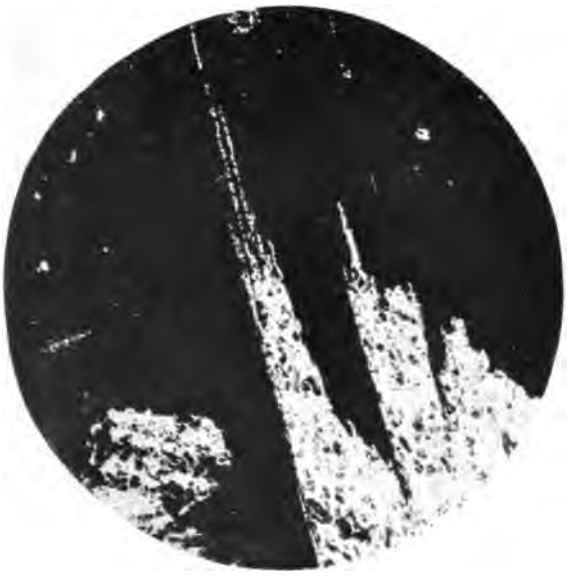


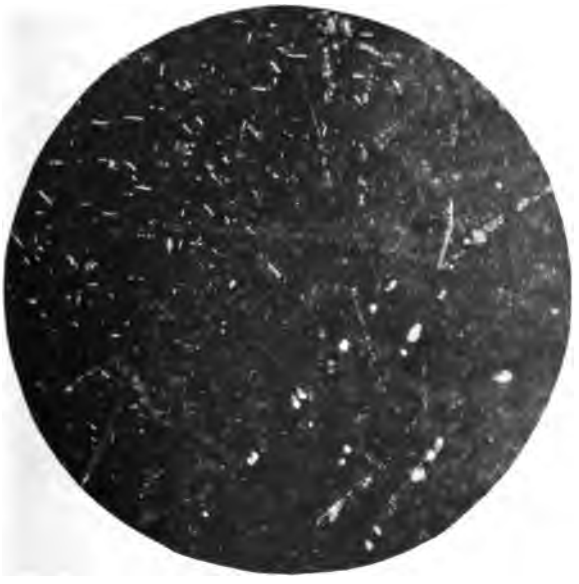
FIGURE 2.—Ore outcrops at Shanton ranch, Wyoming.

METALLOGRAPHIC DESCRIPTION OF ORE.

Etched polished sections of the ore show considerable variation. The ilmenite forms from one-tenth to one-third of the surface of the sections, types of which are shown in Plate II, *D*, and Plate IX, *B*. The ilmenite grains are nearly all about 0.25 mm. in diameter, and some areas of ilmenite have a surface of 104 sq. mm. The shape of the grains varies from approximately equidimensional to distinctly elongate. The elongate grains tend to occur in between the grains of magnetite, and may become as narrow as 0.1 mm. Some of the broader elongated grains fray out rapidly on the ends, as shown in Plate XI, *A*. Though usually following the out-



A. SHANTON RANCH, WYO., ORE. (X 60.)



B. IRON MOUNTAIN, WYO., ORE. (X 60.)

line of the magnetite grains, some of the narrow ilmenite grains take on a regular arrangement within the magnetite.

Except for a few sections in which olivine was present in considerable amount, little gangue is visible. The most prominent mineral besides the olivine is spinel. Though not present in large amount, it occurs throughout the ore in small particles. The most abundant inclusions of ilmenite in the magnetite show on the polished section as dashes and dots. The long lamellæ forming the regular linear network are almost entirely lacking, and their place is taken by the dashes which have an average length of 0.1 mm.

The dashes occur in some cases widely spaced, giving an open structure, as in a part of Plate XI, *B;* in other cases they are very closely crowded, giving a dense network effect as in Plate XII, *A*. This finally becomes so close that the effect of a somewhat continuous surface of ilmenite is produced, such as has already been described in the case of Minnesota and other ores.

The dots are in two sizes, a larger series having a diameter of about $\frac{1}{25}$ to $\frac{1}{50}$ mm. The larger dots occur irregularly distributed, or linearly along outlines of magnetite and rectilinearly parallel to cleavage directions. The irregular distribution of both large and small dots is shown in Plate XII, *B*, and XIII, *A*. The latter also shows the development of the ilmenite lamellæ. The rectilinear arrangement of the larger dots is shown in a part of Plate XI, *B*.

The minute dots occur in irregular swarms and become so minute in places as to be scarcely discernible. A great deal of the magnetite does not show a dull-black surface, but has a gray, mottled appearance. This is doubtless due to just such intergrowths, which have become too minute to be resolved with a microscope. It is shown below that a large percentage of the titanium of the ores occurs within the magnetite, a fact in accordance with the above explanation of the mottled appearance of the magnetite.

A further interesting feature noted is the abundant development of the intersecting lamellæ along certain lines in the magnetite, which seem to be the outlines of the magnetite grains. In such cases the parts of the magnetite at a distance from these lines are practically free from ilmenite lamellæ. This is illustrated in Plate XIII, *B*.

POSSIBILITIES OF UTILIZATION.

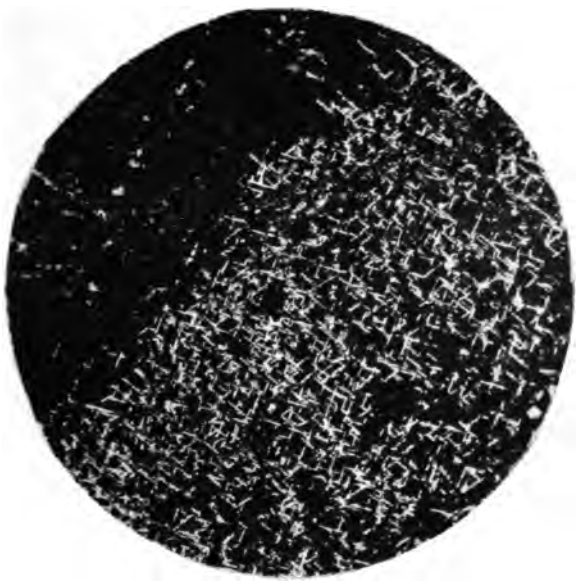
Five specimens of the ore were crushed and separated with a hand magnet. The first three were from Iron Mountain and the last two from Shanton ranch. The results follow:

Results of magnetic separation of Iron Mountain and Shanton ranch ore.

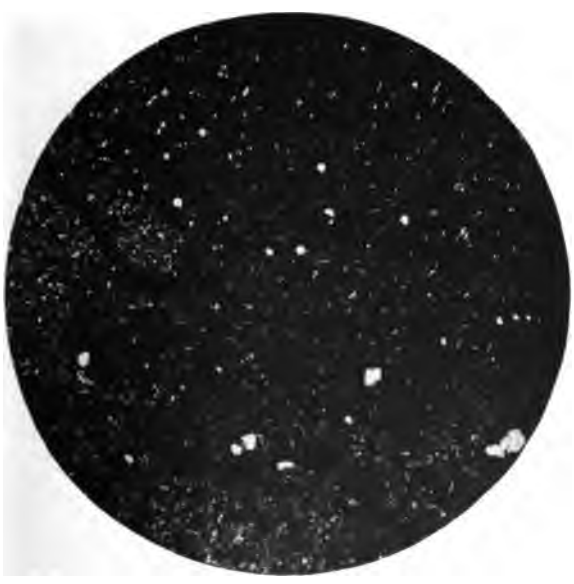
	Unscreened ore.	Ore through 50-mesh, over 100-mesh.			Ore through screen finer than 100-mesh.	
		Magnetic.	Nonmagnetic.	Proportion through screen.	Magnetic.	Proportion through screen.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Proportion	.....	84.2	15.8	48.8	83.6	51.2
Fe	52.31	56.75	28.74	.....	56.75	.....
TiO_2	21.75	18.30	40.12	.....	17.25	.....
Proportion	.....	77.7	22.3	46.3	76	55.7
Fe	54.48	60.76	32.61	.....	61.97	.....
TiO_2	19.47	12.58	43.47	.....	10.32	.....
Proportion	.....	87.3	12.7	46.6	86.5	53.4
Fe	51.93	55.35	28.43	.....	55.44	.....
TiO_2	21.85	20.03	34.33	.....	20.03	.....
Proportion	.....	84.7	15.3	48.0	82.2	52.0
Fe	51.50	55.05	31.83	.....	55.63	.....
TiO_2	21.08	17.85	38.93	.....	17.85	.....
Proportion	.....	84.1	15.9	47.5	78.6	52.5
Fe	52.98	56.93	31.96	.....	57.30	.....
TiO_2	19.98	17.22	34.58	.....	16.72	.....

Of these five samples the average Fe content is 52.64 per cent, and the average TiO_2 content, 20.83 per cent. Of the 50-100 mesh concentrate the average Fe content is 56.97 per cent, and the average TiO_2 content, 17.20 per cent. Of the concentrates finer than 100-mesh the average Fe content is 57.42 per cent, and the average TiO_2 content, 16.43 per cent. The average composition of the 50-100 mesh tailings is Fe 30.71 per cent, TiO_2 38.29 per cent. This corresponds to a composition of the original ore of 52.56 per cent magnetite and 39.58 per cent ilmenite; of the 50-100 mesh concentrates, 62.05 per cent magnetite and 32.68 per cent ilmenite; of the concentrates that passed through a 100-mesh screen, 63.41 per cent magnetite and 31.22 per cent ilmenite; of the 50-100 mesh tailings, 72.75 per cent ilmenite and 5.40 per cent magnetite. It is again seen that there is little advantage in grinding finer than 50-mesh. This is what one could predict from the size of the ilmenite grains, as practically all are larger than the size passing through a 50-mesh screen, and the ilmenite intergrowths in the magnetite are too minute to be separated by the 100-mesh crushing.

The disappointing feature is the small amount of titanium removed. A careful study of the ore, however, reveals the cause of this. Polished sections of the five specimens of ore subjected to magnetic



A. IRON MOUNTAIN, WYO., ORE. (× 60.)



B. SHANTON RANCH, WYO., ORE. (× 60.)

A. IRON MOUNTAIN, WYO., ORE. (× 60.)

B. IRON MOUNTAIN, WYO., ORE. (× 60.)

A. IRON MOUNTAIN, WYO., ORE. (× 60.)

B. IRON MOUNTAIN, WYO., ORE. (× 60.)

nental Divide. Its elevation above tide is 9,900 feet. Two small openings were made at the western end of the outcrop some years since, and a little ore was taken out and teamed to Boyd's smelter at Boulder. The pits are 8 to 10 feet deep and 6 to 8 feet wide. The ore exposed in them and seen on the dump varies greatly in richness, the mass being made up of little seams of rich, compact magnetite, lying in a siliceous magnetic rock. The outcrop extends along the crest of the hill, and is about 400 feet in length. The north side of the hill is completely covered with float ore. All the ore is very magnetic, and exhibits polarity to so marked a degree that the dip of the needle changes from + 90° (north end down) to − 50° (south end down) in the distance of 10 paces. There is undoubtedly a large amount of iron ore in the hill, but it is probable that the rich ore is too irregularly distributed through the rocks to permit of its being economically mined, even if other circumstances were favorable.

Chauvenet[a] spoke of the deposit in 1886 as a vein of black magnetite, and gave an analysis of the ore which is cited below. In 1890 he made the following mention[b] of the occurrence:

Beginning with Boulder County, we have the vein in Archean rock already mentioned, which is found on Caribou Hill, close to many rich silver veins, though not itself forming part of the system of veins which carry the precious metals. This vein powerfully affects the compass in its vicinity and is called a deposit of magnetite, though, as far as analysis is concerned, it might be named ilmenite, since it contains the unusual amount of 36 per cent of titanic acid. Its percentage of iron is from 36 to 38. The ore is compact, jet black, and, of course, worthless as iron ore.

Kemp[c] also refers to the deposit briefly, saying that beyond establishing the character of the ore little has been accomplished and no determinations of the wall rock have been recorded.

A detailed account of the deposit has recently been published by Jennings.[d]

GENERAL FIELD RELATIONS.

The deposit lies in the northeastern corner of the Central City quadrangle, which has recently been mapped by E. S. Bastin and J. M. Hill, of the United States Geological Survey. A detailed description of the geology and the ore deposits of this district is now in the course of preparation by the Survey and only a brief sketch of the geology is given here.

The district is within the belt of pre-Cambrian granites and gneisses forming the Front Range. These rocks have been intruded by numerous dikes ranging in composition from acidic to extremely basic. On Caribou Hill are several intrusions of basic material, and it is within these that the ore occurs. The most important one is the dike on the northeast slope of the hill, which can be traced for over 1,500 feet. It begins at a point 500 yards west of Caribou and

[a] Chauvenet, Regis, Notes on iron prospects in northern Colorado: Biennial Rept. of the Colo. State Sch. of Mines, 1886, p. 16.

[b] Chauvenet, Regis, the iron resources of Colorado: Trans. Am. Inst. Min. Eng., vol. 18, 1890, p. 267.

[c] Kemp, J. F., A brief review of the titaniferous magnetites: Sch. of Mines Quart., 1899, p. 335.

[d] Jennings, E. P., A titaniferous iron-ore deposit in Boulder County, Colo.: Bull. Am. Inst. Min. Eng., October, 1912, pp. 1045–1056.

runs up the hill slope in a southeasterly direction for about 400 feet, where it bends sharply to the east and continues for about 1,000 feet to the edge of Caribou. A second intrusion, having more of a stock shape, occurs on the southeast slope of the hill, half a mile southwest of Caribou. A third smaller outcrop occurs near the top of the hill about 100 yards southeast of the summit. These rocks show considerable local variations, but are essentially pyroxenitic in character. Associated with the pyroxene in places is feldspar and biotite. The texture ranges from that of a coarse pyroxenite to a very fine-grained rock.

THE ORE DEPOSITS.

Some magnetite occurs as an accessory constituent of the basic rocks, and this becomes segregated into richer pockets, forming a medium-grade ore. But even in the ore-bearing parts of the rock the composition is not uniform. It is rather a rock rich in magnetite with numerous stringers and veinlets of almost pure magnetite. The magnetite is highly magnetic and much of it is natural lodestone. These stringers of pure ore never exceed 6 inches in thickness and seldom measure more than an inch or two. They may form such a network within the leaner rock as to constitute a medium-grade ore. Such ore bodies do not reach a width of more than a foot or two or a length of more than a few feet. The ore in the pits and trenches that have been made along the western end of the dike northwest of Caribou consists of a series of closely crowded lenses of that character. The dike has a width of 50 to 100 feet or over, but only a small part consists of ore of even this grade. The ore in the other two occurrences is still leaner and there is still less of it. Thin sections of the ore show that it consists essentially of the ore minerals and pyroxene, the latter ranging in composition from diopside to augite. The pyroxene is frequently altered to serpentine, and in part to hornblende. Some biotite is also usually present. Inclusions of dark-green spinel in the magnetite are common.

The following analyses of the ore have been published. Analyses 1 and 2 are taken from the Tenth Census Report and analysis 3 from Chauvenet. Analysis 4 is calculated from the more detailed analysis by Jennings, which is also given in full.

Results of analyses of Caribou iron ore.

Constituent.	1	2	3	4
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Fe	59.04	35.35	36.93	50.97
SiO_2	.....	.....	1.16	9.60
TiO_2	Present.	Present.	36.76	4.48
P	.017	.022	.04	.....

Mineralogical composition of medium-grade ore from northwest dike.

[Analyst, E. P. Jennings.]

	Per cent.
Ilmenite	8.50
Magnetite	67.25
Apatite	.18
Spinel	6.70
Diopside	16.89
H_2O at 105°	.51

METALLOGRAPHIC DESCRIPTION OF ORE.

A metallographic study of the ore shows little of interest. The richer veinlets consist of medium-grained aggregates of magnetite and ilmenite with a little gangue. The ilmenite is far less abundant than the magnetite and occurs in smaller grains, in many places as small particles filling interstices between the magnetite and the gangue grains. The leaner ore, which grades over into the country rock, contains scattered grains of magnetite and ilmenite, with the femic minerals in excess.

Intergrowths of ilmenite in the magnetite are not abundant. Most frequently they consist of dots and small particles scattered irregularly through the magnetite. Regular intergrowths of tabular ilmenite in the magnetite are not numerous. Altogether, the quantity of ilmenite seen in polished surfaces would not lead one to expect the high titanium content given by Chauvenet in his analysis, a conclusion that is corroborated by the analysis by Jennings presented above.

POSSIBILITIES OF UTILIZING ORE.

Titaniferous magnetite occurs in small pyroxenitic intrusions on Caribou Hill. The best of the ore is only medium grade and the ore lenses are very small. On account of its small size the deposit can never have any economic value.

IRON MOUNTAIN.

Iron Mountain is situated in the southern part of Fremont County, about 12 miles southwest of Cañon City, and 15 miles north of Silver Cliff. It lies on a branch of Pine Creek, which is itself tributary to Grape Creek, and the locality is often referred to as Grape Creek. More exactly, it is 3½ miles east of the western edge and 1 mile north of the 38° 20′ line of the Cañon City topographic sheet.

Though the deposit has been visited several times, there is little information to be had in regard to it. This is doubtless the deposit referred to by F. M. Endlich in 1873.[a] He says: "A short distance up Grape Creek Cañon a deposit of magnetic iron ore occurs, which

[a] Hayden, F. V., Seventh Annual Report of the United States Geological and Geographic Survey of the Territories, 1873, p. 333.

is being worked at present." This was probably about the beginning of mining operations here, as Putnam[a] states that a United States patent for the land was obtained in 1872. During this period the ore was hauled by wagon to Cañon City by freighters on their return trips to that point. Shortly before 1880 a branch line of the Denver & Rio Grande Railroad was built up Grape Creek from Cañon City to Silver Cliff. This passed within 4½ miles of the deposit, and preliminary surveys were made for a branch line up Pine Creek to the deposit, but the line was never built; and in 1888, after a series of washouts, the Silver Cliff line was abandoned. The deposit was worked at several points, but none of the pits are large, the average being about 50 feet long, 10 to 15 feet wide, and 10 to 15 feet deep. Consequently the total amount of ore mined was not great.

The most complete account of the deposit is given by Putnam,[b] and in his report a number of analyses of the ore are included.

In 1885 Chauvenet,[c] referring to these ores, says:

> The ore is a magnetite, not very siliceous, but containing from 10 to 15 per cent of titanic acid. Its use, even in relatively small quantity, was found extremely troublesome, and was limited to such a proportion that the entire mixture of ores charged should contain but 1 per cent of the impurity. Even at this low figure the working of the furnace was retarded. This ore yielded 45 per cent of iron and sometimes more, but the workings have been practically abandoned.

Again, in 1890, he says:[d]

> In Fremont County is the mine known as the Grape Creek. The ore is titaniferous, showing a very steady average of about 14 per cent of titanic acid. Its iron runs about 48 per cent. It is no longer used in the blast furnace, but is useful as "fix" in the puddling mill. The deposit is a large one, and it is a pity that its character prevents its more extensive use.

Kemp's account[e] is based chiefly on Putnam's, but in addition he determined a specimen of gray granitoid rock in a collection of the titaniferous ores at Columbia University to be olivine gabbro. This is the first and only information we have as to the character of the rock associated with the ore.

GENERAL FIELD RELATIONS.

The prevailing rocks of the region are granites and gneisses. These have been cut by numerous dikes, prominent among which are pyroxenite. Besides the dikes, they have also been intruded by a large mass of gabbro. The greater part of this mass forms a high hill north of the ore deposits, which is known as Iron Mountain.

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[a] Putnam, B. T., Tenth Census, vol. 15, 1886, p. 472.

[b] Op. cit., pp. 272–276.

[c] Chauvenet, Regis, Preliminary notes on the iron resources of Colorado: Ann. Rept. of the Colo. State Sch. of Mines, 1885, p. 16.

[d] Chauvenet, Regis, The iron resources of Colorado: Trans. Am. Inst. Min. Eng., vol. 18, 1890, p. 270.

[e] Kemp, J. F., A brief review of the titaniferous magnetites: Sch. of Mines Quart., 1899, pp. 333–334.

The gabbro continues southward across a low flat spur of Iron Mountain and two small hills separated by dry arroyas. It is on this spur and the two hills that the ore deposits occur. The ore-bearing part of the gabbro takes on a more feldspathic character than that composing Iron Mountain, and in places grades over to a true anorthosite.

The gabbro is likewise cut by dikes, the most common of which are hornblende syenite with accessory zircon and magnetite. At the south end of the spur of Iron Mountain are several dikes of a dioritic rock. This location is marked A on the map constituting figure 3. One of the dikes cuts through both the gabbro and an ore segregation in it. The groundmass of the rock is a coarse granular aggregate of plagioclase feldspars of about the composition of oligoclase. Zircon occurs as an accessory constituent, and in one part of a slide is a swarm of minute isotropic grains which seem to be spinels. Green pyroxene and amphibole are prominent constituents of the rock. The crystals of these minerals are usually larger than those of the feldspar, and some of the hornblende prisms attain a length of 3 cm. and a diameter of 0.5 cm.

As stated above the gabbro varies in character from a typical olivine gabbro to one in which the ferromagnesian minerals become accessory and which approaches a typical anorthosite in composition. The most abundant ferromagnesian mineral in the gabbro is augite; less abundant but prominent is olivine. Magnetite, with numerous inclusions of green spinel, and a little biotite and hornblende are also present. The anorthositic phases contain the same minerals in subordinate amounts. A slide of this rock shows the olivine surrounded by rims of hornblende so that it usually does not come in contact with the feldspar.

THE ORE DEPOSITS.

The first hill south of the spur from Iron Mountain affords the most promising showing of ore, and it is here that most of the openings were made. This hill is 800 feet long, 500 feet wide, and 60 feet high. The locations of the principal ore bodies in it were determined by Putnam by means of a magnetic survey, and are shown in figure 3, reproduced from his map. These ore bodies are exposed at several points both in natural exposures and in the old ore pits, and the relations of the ore to the wall rock are well shown.

The distinction between what is ore and what is wall rock can generally be sharply drawn. The wall rock may be anorthosite or it may be a gabbro rich in ore minerals; the ore may be a rich magnetic ore with little gangue, or it may be a lean ore carrying large percentages of the common constituents of the adjacent wall rock. But no matter how closely the two may approach each other in composition, there

hardly ever exists what one would call a gradual transition. The change from wall rock to ore is so rapid that one has no difficulty in fixing a contact.

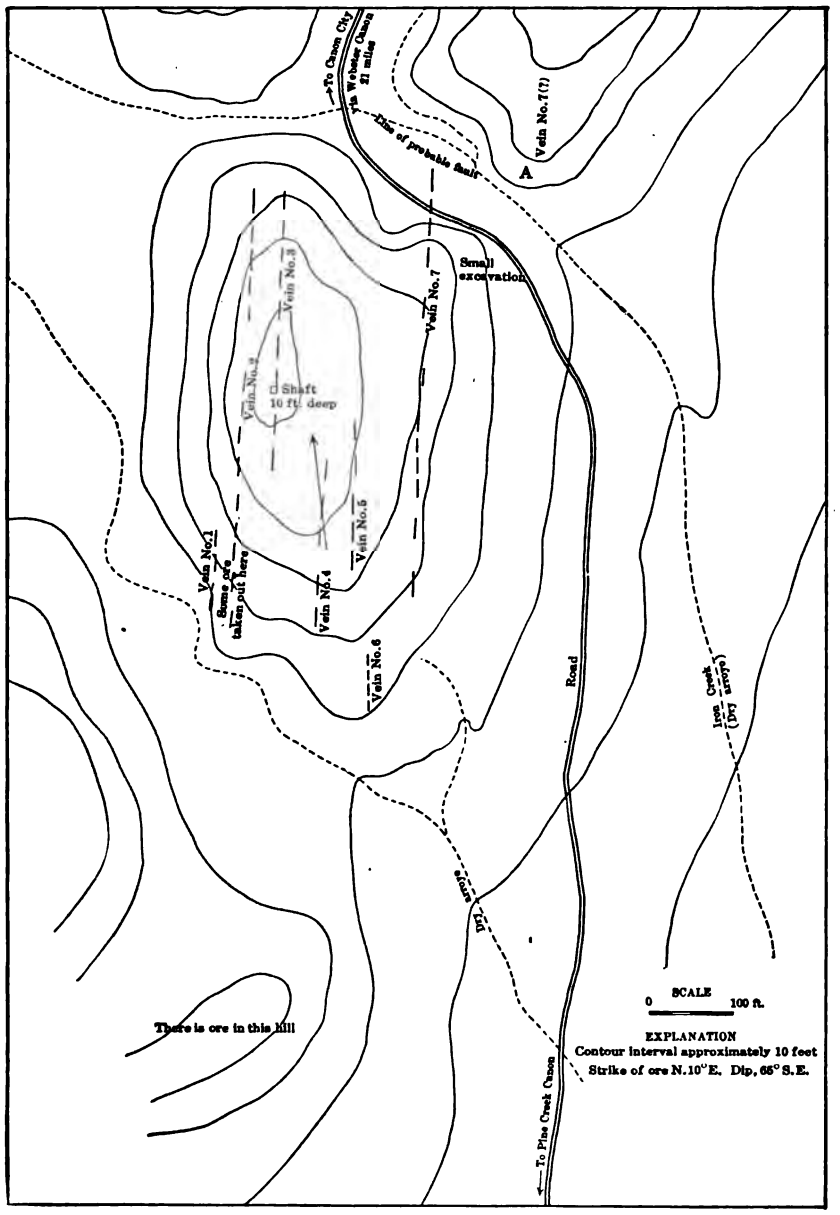


FIGURE 3.—Map of Iron Mountain, Colo. [From Tenth Census report.]

The width of the ore bodies ranges from about 10 to 50 feet, and does not average more than 25 feet. Their length can not be determined on account of lack of exposure. Whether those indicated on

Putnam's map (fig. 3) are continuous or represent zones of ore lenses is an open question. The latter supposition is the more probable.

The ore itself is rather uniform in composition and appearance. It is a compact aggregate of magnetite and ilmenite, on fresh fractures of which the ilmenite can be recognized by its higher luster. The gangue minerals are the constituent minerals of the gabbro. In some specimens the gangue is very prominent; in general, however, the ore is of good grade. The most characteristic mineral of the gangue is the feldspar. It has a tabular habit, and tends to a subparallel arrangement, which gives to the ore a striking diabasic appearance. The similarity of this ore to some of the Minnesota ore has already been mentioned. Plate VIII, *C*, shows a specimen of the ore in which the white feldspar crystals are plainly visible against the dark background of the ore minerals. A thin section of the ore consisted of nearly three-fourths magnetite in which are abundant inclusions of dark-green spinel. The most prominent gangue minerals were the lath-shaped feldspars. Olivine was less abundant and biotite only sparingly present. Along the boundary of the feldspar and olivine against the magnetite is a narrow rim of a yellowish-brown mineral which seems to be hornblende.

Several analyses of the ore have been made and they show a very uniform composition. The analyses are tabulated below:

Results of analyses of Iron Mountain ores:

Constituent.	1	2	3	4	5	6
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Fe	48.89	48.18	46.61	48.85	46.05	47.01
TiO_2	11.99	12.92	11.61	12.73	13.84	14.86
P	.025	.026	.037	.011	.040	.039

Constituent.	7	8	9	10	11
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Fe	47.42	49.35	43.4	48.79	51.96
TiO_2	13.04	14.00	10.0	.....	14.5
P	.026	.....	.....	Trace.	.....

Constituent.	8	9	10
	Per cent.	*Per cent.*	*Per cent.*
Silica	8.60	18.60	2.75
Alumina	4.00	7.00	7.70
Lime	1.60	1.30	None.
Magnesia	2.30	1.60	3.20
Titanic acid	14.00	10.00	.....
Peroxide of iron	70.50	62.00	.....
Magnetic oxide of iron	.....	.....	67.76
Sulphuric acid	.....	.....	Trace.
Phosphoric acid	.....	.....	Trace.
Oxide of manganese	.....	.....	Trace.
	101.00	100.50	.....

Analyses 1 to 7 are taken from Putnam's report, analyses 1 to 5 representing samples taken from the ore bodies indicated on figure 3 by the numbers 2, 3, 6, 3, 4, respectively. The sources of samples represented by analyses 6 and 7 are not given. The samples represented by analyses 8 and 9 were collected by M. Chaper, and the analyses made at the School of Mines, Paris. M. Chaper, of the Geological Society of France, visited the deposit in 1875, and extracts of his report together with the analyses were published in 1877–78. The analyses as here presented are taken from Putnam's report, in which they are cited. The sample represented by analysis 8 is described as a "compact titaniferous magnetite with feldspathic gangue"; the sample represented by analysis 9 is described as "an oxidized iron sand, forming the bed of Iron Creek for several miles above Iron Mountian. A portion of the iron exists in the state of protoxide, which is all converted into peroxide, whereby the increase in weight results." Analysis 10 was made in 1871 by Charles S. Hinchman and was quoted by Putnam. Analysis 11 was calculated from analyses of concentrates and tailings made by A. C. Fieldner, of the Bureau of Mines.

The average composition of the ore is shown to be 47.86 per cent Fe, 12.95 per cent TiO_2, and the phosphorus content considerably below the Bessemer limit. The titanium content is about the average for titaniferous magnetite.

METALLOGRAPHIC DESCRIPTION OF ORE.

All the polished sections of the ores that were studied are much alike. On all of these the gangue constitutes less than one-fourth of the surface and averages about one-eighth. Ilmenite grains seldom exceed 10 per cent of the surface, and the average is below 10 per cent. Manifestly, the magnetite must contain a considerable part of the titanium.

A conspicuous feature of the magnetite is the almost complete absence of the large intersecting lamellæ of ilmenite. The most common ilmenite intergrowths are dots, which occur in great numbers. Though every intermediate size occurs, they tend to be of two orders of magnitude. The most abundant are extremely minute dots, examples of which have already been shown in the case of other deposits. (See Pl. XII, *B*, and Pl. XIII.) The larger series have a diameter which may be expressed in hundredths of a millimeter, and in places become as great as 0.1 mm. Both sizes of dots may be, and usually are, present in the same specimen. Pl. XIV, *A*, shows the large dots both irregularly distributed and in linear arrangement.

On some surfaces are seen extremely delicate networks of closely spaced short lamellæ in the midst of which are dots of the larger size. The effect is that of a minute network upon which the dots

have been superimposed. This is illustrated in Plate XIV, *B*. Such a network in many places grades over into a more minute network until finally the intersecting structure can no longer be recognized even under high power. The transition is gradual enough to make certain that the disappearance is not due to the discontinuance of the network, but to the fact that it becomes so delicate as not to be discernible.

Another method of intergrowths is the occurrence of well-formed laths of ilmenite. These range from 1 to 2 mm. in length and may have a width of as much as 0.2 mm., though usually much less. This feature is shown only in a few places. It reminds one of the intergrowths of the North Carolina ore shown in Plate VII, *A*. In this case, however, the laths are smaller and much more regularly formed.

POSSIBILITIES OF UTILIZING ORE.

The structure of the ore revealed on polished surfaces is such as to make impossible the elimination of any considerable percentage of the titanium by means of magnetic concentration. Not more than one-third of the titanium occurs in grains large enough to be isolated when crushed to 100-mesh. The following results of a test on a specimen of the ore are even more disappointing than one would have expected:

Results of magnetic separation of Iron Mountain ore.

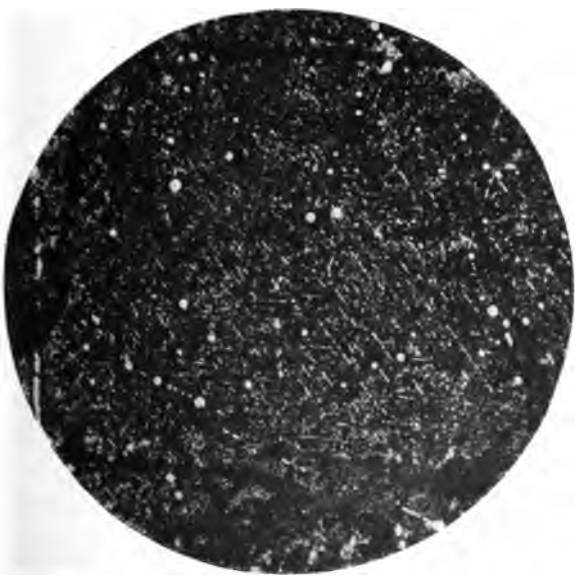
	Ore.	Ore that passed through 50–100 mesh screen.[a]		Ore that passed through screen finer than 100-mesh.[b]
		Magnetic.	Nonmagnetic.	Magnetic.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Quantity	.....	86	14	80.4
Fe	51.96	56.93	21.42	57.77
TiO_2	14.5	13.5	20.97	13.1

[a] 46 per cent. [b] 54 per cent.

Here as in other cases the separation is little better in the more finely crushed ore, owing to the fact that the ilmenite grains are nearly all larger than a 50-mesh opening and the intergrowths in the magnetite are nearly all smaller than a 100-mesh opening.

The ores have about the average composition of titaniferous magnetites, but the titanium is intergrown with the magnetite in such a manner as to make possible only a very incomplete separation of the two. Hence their utilization depends entirely upon the possibility of using ores containing 14 per cent TiO_2. But even in such an event

A. IRON MOUNTAIN, COLO., ORE. (X 60.)



B. IRON MOUNTAIN, COLO , ORE. (X 60)

the deposit is not very promising. The individual ore bodies are not large and the total tonnage not great; also, the expense of securing proper transporation facilities would be heavy. Consequently this occurrence can not be looked upon as an important ore reserve.

CEBOLLA CREEK.

The Cebolla district is located on the east side of Cebolla Creek, south of Powderhorn, in the southern part of Gunnison County. It lies about 9 miles east of the Lake City branch of the Denver & Rio Grande Railroad, but is best reached by a stage line from Iola, on the Denver & Rio Grande narrow-gage line which lies 15 miles to the north.

The occurrence of titaniferous magnetite in this region has been known for some years, but no description of the deposit has ever been published.

J. K. Hallowell [a] visited the deposit in 1883, and said of it:

> The Cebolla iron deposit is a peculiar formation unto itself. I was asked regarding its geology. I had to weaken; I never saw anything like it or read of anything similar. There is iron enough, over a mile square of it, apparently overlaid by trachyte at one time. The country rock is granite. The ore is heavy black magnetite, of fair grade of iron, but carrying, as I was afterwards informed, * * * too much titanium to be practically available in iron manufacture.

Chauvenet [b] in a report on the iron resources of Gunnison County in 1887 makes the following mention of it:

> We come next to a region known as the "Cebolla" district, situated near White Earth Creek (usually called Cebolla Creek) and at an average distance of about 26 miles from Gunnison City. This district lies very close to a proposed extension of the Denver & Rio Grande Railroad to Lake City, Hinsdale County.
>
> An immense ledge of titaniferous iron ore exists here, but little need be said of it. Samples taken from it vary greatly in their percentage of this objectionable material, the lowest I have ever analyzed yielding 9.38 per cent and the highest 36 per cent. This last was rather to be classed as "ilmenite" than as iron ore. This deposit is enormous in extent and is well worth exploitation with a view to ascertaining whether in parts or at depth it will not yield a product lower in titanium than any yet discovered. At present it must be condemned as unfit for smelting, at least by itself.

Two years later Chauvenet again refers to the deposit as follows: [c]

> Next, as a curiosity, not as a commercially valuable mine, I would name the Cebolla Creek titanium deposit. No one knows how the ore will hold out here, but as it can be developed in many places for some miles it will last, in all likelihood, far longer than there is any use for it. It contains from 9 to 36 per cent of titanic acid. We have all read in our textbooks on chemistry that titanium is a "rare" metal. I do not think I risk much in saying that this is an inexhaustible stock of it.

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[a] Hallowell, J. K., Gunnison, Colorado's Bonanza County: Colo. Mus. App. Geol., 1883, p. 106.

[b] Chauvenet, Regis, Iron resources of Gunnison County: Ann. Rept. State Sch. of Mines of Colo., 1887, p. 18.

[c] Chauvenet, Regis, The iron resources of Colorado: Trans. Am. Inst. Min. Eng., vol. 18, 1890, p. 272.

In 1896, Arthur Lakes [a] described deposits of nontitaniferous iron ore which occur in the immediate vicinity of the titaniferous iron, but made only a brief reference to the latter. He examined the small opening close to the road, directly opposite the Cebolla Hot Springs, and described the ore as "gray massive iron" which is "mingled with streaks of brown mica and a green, somewhat asbestiferous-looking mineral, which may be hornblende or pyroxene."

Kemp [b] refers to the deposit and comments on the lack of information in regard to the ore or wall rock.

In the summer of 1905 or 1906 the district was visited by Van Hise and Leith, [c] and is classed by them genetically with the Iron Springs district of Utah; that is, as a contact metamorphic deposit.

The above information in regard to this district, though meager, was of such a character as to excite considerable interest from a scientific point of view. No titaniferous magnetites of contact metamorphic origin are known to the author. Krusch [d] cites the absence of titanium as one of the characteristics of these ores, and Beyschlag, Krusch, and Vogt [e] state that experience up to the present has been that these ores are characterized by an almost complete absence of titanic acid, with an occasional trace.

Here was seemingly a highly titaniferous contact metamorphic magnetite deposit, and the author's examination of the same showed that such actually was the case for a part of the deposit. A detailed account of the geological features of the deposit has recently been published by him [f] and the geological discussion in this report is restricted to a summary of that account.

GENERAL FIELD RELATIONS.

The Cebolla district lies within a region of pre-Cambrian rocks consisting mostly of granites and gneisses. In the midst of these rocks is an area of basic igneous rocks, limestone, and rhyolitic lavas. It is in association with the basic igneous rocks and the limestone that the titaniferous iron ores occur. The basic rock contains numerous small segregations of ore. It has intruded a part of the limestone in a most intricate manner, producing products of contact metamorphism and the deposition of contact metamorphic iron ores.

[a] Lakes, Arthur, Iron and manganese; The great Cebolla River deposits: Coll. Eng. and Metal Miner, 1896, pp. 267–268.

[b] Kemp, J. F., A brief review of the titaniferous magnetites: Sch. of Mines Quart., 1899, p. 335.

[c] Leith, C. K., and Harder, E. C., The iron ores of the Iron Springs district, southern Utah: U. S. Geol. Survey Bull. 338, 1908, pp. 92, 93.

[d] Krusch, P., Die Untersuchung und Bewertung von Erzlagerstätten, 1907, p. 179.

[e] Beyschlag, Krusch, Vogt, Die Lagerstätten der nutzbaren Mineralien und Gesteine, vol. 1, 1909, p. 351.

[f] Singewald, J. T., jr., The iron-ore deposits of the Cebolla district, Gunnison County, Colo.: Econ. Geol., 1912, pp. 560–573.

ORE DEPOSITS.

MAGMATIC SEGREGATIONS.

The basic igneous rock has undergone considerable differentiation, and shows rapid variation from place to place, especially in the ore-bearing area. To the north of the ore-bearing area it consists of a dark, greenish, fine-grained rock, in which locally considerable chalcopyrite is disseminated. The leaching of this has given rise to green, copper stain in crevices of the rock, and at one such point some prospecting has been done. Within the ore-bearing area the rock is somewhat coarser grained, and especially in association with the ores becomes very coarse grained. In general the coarse-grained varieties are feldspar poor or free, though typical feldspar-rich gabbro does occur. Most of these coarse varieties consist of coarsely crystalline aggregates of pyroxene, biotite, and ore in all possible ratios.

The richer ores consist of aggregates of titaniferous magnetite and bunches of dark-brown mica which reach several centimeters in diameter. In such specimens pyroxenes are not abundant. They grade over, however, into leaner ores in which the pyroxenes are more abundant, until a rock results that consists almost entirely of pyroxene. The ores are not entirely confined to these coarse rocks, but also occur as veinlets and stringers in the adjacent fine-grained rocks. These ores were prospected and worked to a slight extent about 30 years ago by a Mr. Crooks, of Lake City, who hauled some to the smelter located there at that time. Their titaniferous character was soon discovered and further shipments were refused by the smelter. The most extensive openings were made in a draw 1 mile northeast of the Hot Springs. Another opening 50 feet long, was made just above the main road in the valley due east of the Hot Springs. Besides these, numerous smaller openings have been made, especially where the intrusions in the limestone occur. The ores throughout the area show the same features, both as to their mineralogical associations and their geological occurrence. There is no evidence of the occurrence of any large ore bodies, but rather of a large number of very small ore segregations. No analyses of the ores were made, but the metallographic study of the ores corroborates the high percentages of titanium given by Chauvenet. In almost all specimens examined the ilmenite was far in excess of the magnetite.

The composition of the ores, their mineralogical association, and their manner of occurrence establish beyond doubt their genetic position. They are magmatic segregations of titaniferous magnetites in a basic igneous rock.

CONTACT METAMORPHIC ORES.

Petrographically the area within which the limestone has been intruded by the basic rock affords a complete gradation from the basic igneous rock to a pure-white, coarsely granular marble. In some places the igneous rock has resorbed some of the limestone, so that calcite occurs in it as a primary constituent. At other places occurs limestone in which large mica flakes are abundantly disseminated. Elsewhere, typical products of contact metamorphism of limestone have been formed. The transitions are so rapid that one can pass from one extreme to the other within a few paces. The effect is that of a limestone mass that has been intricately intruded by the igneous. Sufficient time was not available to determine whether the intrusion took place in the form of a large number of distinct dikes, or in the form of irregular stocks. The complex is the least resistant of the rocks of the region, and hence has not afforded many pronounced outcrops.

The igneous rock is the same as that described above, and associated with it are small ore segregations of the character described.

The most interesting feature of this complex is the occurrence of small deposits of contact metamorphic ore. The product of contact metamorphism is usually a light-green, compact rock. In thin section, this rock is seen to be made up of calcite, augite, garnet, and, less abundantly, of zoisite and vesuvianite. The relative abundance of these minerals varies greatly in different specimens. In many places there occurs, as phenocrysts as much as 5 centimeters long, a dark augite with purplish tinge. Less abundant than this light-green, dense rock is a somewhat coarser dark rock which contains a larger percentage of yellow to brown garnets and deep-colored augite. Within the contact metamorphic rock are numerous pockets and nests of magnetite. These pockets of ore, though small, contain very little gangue. They are usually separated from the surrounding light-green rock by a narrow zone of the darker coarse-grained mixture of garnet and augite. A study of polished sections of this ore, one of which is reproduced in Plate II, *F*, showed that it consists of both ilmenite and magnetite; and, though seemingly not as abundant in amount as in the magmatic segregations, the ilmenite forms a considerable part of the ore. From a scientific standpoint, this is the most important and striking feature of the district—the occurrence of highly titaniferous iron ore of contact metamorphic origin.

REPLACEMENT ORES.

A third type of ore in the district is a replacement of the limestone, which, as it is not titaniferous, is of no particular interest in this connection, though the ultimate source of the iron was probably the same in all three types.

METALLOGRAPHIC DESCRIPTION OF THE ORE.

Though the individual segregations are small, the ore itself is rather high grade, and polished sections contain little gangue. The average percentage of gangue on a number of polished surfaces of the ore scarcely exceeds 15 per cent. The ore grains average from 2 to 5 mm. In some cases they reach as much as 10 mm. and few are smaller than 0.5 mm. The ratio of ilmenite to magnetite varies between wide limits in different specimens but is uniformly great, indicating an ore high in titanium. No analyses were made of this ore, but Chauvenet gives the range of TiO_2 between 9 and 36 per cent. In only a few sections did the magnetite exceed the ilmenite; and in some cases the ilmenite made up more than three-fourths of the surface. A very common relation of ilmenite to magnetite is that illustrated in Plate II, *E*. The two minerals, magnetite and ilmenite, are of so nearly the same hardness that no differences in relief are brought out by the ordinary amount of polishing. Two specimens (Pl. II, *E* and *F*) of this ore, however, which were polished longer than usual show a slight difference. The highly polished surface of the ilmenite has been cut back a little so that the magnetite stands out in slight relief.

The inclusions of ilmenite in the magnetite present several features of interest. One of the larger features is the segregation of irregular elongated particles of ilmenite about the periphery of ilmenite grains. These particles are not in direct contact with the ilmenite, but are separated from it by a narrow zone of magnetite free from ilmenite. This is well shown at the right end of Plate II, *E*. The effect is different from that described in the case of the Adirondack ore and illustrated in Plate VI, *B*. In this case the elongated particles of ilmenite are arranged parallel to the outline of the large grains, whereas in the case of the Adirondack ore they tend to orient themselves at right angles to them. Hence the broad, somewhat continuous bands formed by the coalescing of the ilmenite particles in the Adirondack ore are not found in the Colorado ore.

Many of the sections showed a tendency for the ilmenite to occur as a thin layer between the individual magnetite grains, so that their outlines are marked by a bright line of ilmenite, as has been illustrated in Plate IV, *A*. In places, instead of the simple film of ilmenite, there is a marked development of lamellæ along the outline as is seen in Plate XV, *B*. Very similar intergrowths were observed in some of the Iron Mountain, Wyo., ores, and are shown in Plate XIII, *B*.

A network of large intersecting lamellæ, as in Plate XV, *B*, is not very common, but is in places unusually well developed, the lamellæ attaining a length of 2 mm. In one section, characterized

by an unusually heavy network, the thickness of the lamellæ was 0.07 mm. In most cases the network is poorly developed; the lamellæ occur in only a part of the magnetite crystals, only one series being prominent, or they are extremely short and faint. The short lamellæ are particularly well illustrated in a section of the contact metamorphic ore reproduced in Plate XVI, *A*.

The most common form of ilmenite inclusion is as minute dots. Many of the magnetite grains are swarming full of such minute particles of ilmenite. As a rule they seem to be distributed without any regularity, but in places they tend to have a linear arrangement. This may even become sufficiently pronounced to give the effect of a triangular network of dotted lines. In some cases the dots are slightly elongated, and the network made still more striking. It would seem therefore that the swarms of minute dots represent the incipient stage in the formation of the network of intersecting ilmenite lamellæ. The linear arrangement of the dots on the polished surfaces represents the orientation of the dots into planes parallel to the octahedral faces of the crystallizing magnetite. The elongation of the dots is caused by the coalescing of several dots. As this process continues small lamellæ of ilmenite are formed in intersecting series of parallel plates.

POSSIBILITIES OF UTILIZING ORE.

The possibilities of these ores are not very promising. Although a considerable tonnage of ore doubtless exists, it occurs in the form of countless small segregations, so that large-scale operations are impossible. Magnetic concentration to obtain a concentrate low in titanium is not feasible, because ilmenite forms more than 50 per cent of the ore, so that the percentage of magnetite concentrate obtained would be very small. The inaccessible location of the deposit is also a serious drawback.

RÉSUMÉ.

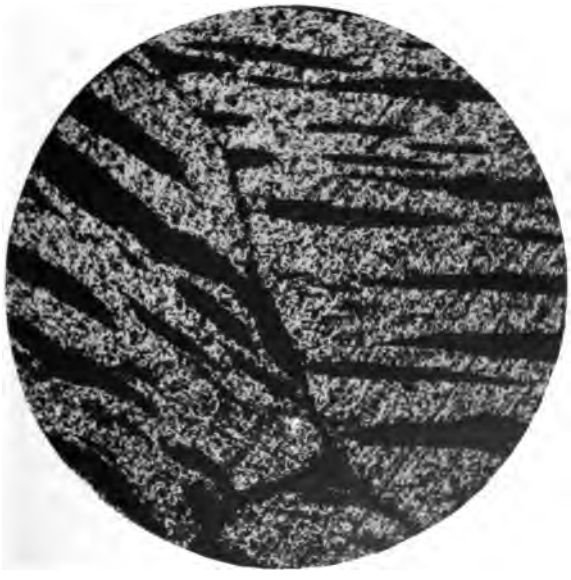
The evolution of the modern blast furnace has taken place in such a direction as to make the utilization of titaniferous iron ores impracticable. Notwithstanding numerous experiments that have been conducted with a view to discovering a furnace charge that will make the use of these ores practicable from the standpoint of furnace operation and economy, there is to-day no hopeful feeling in regard to the possibilities of smelting these ores in the blast furnace. On the other hand, the introduction of the electric furnace holds out a new hope for the direct reduction of the titaniferous ores.

The results of magnetic-separation experiments conducted with these ores are very promising in some cases and extremely disap-

A. CEBOLLA CREEK, COLO., ORE. (× 60.)

B. CEBOLLA CREEK, COLO., ORE. (× 60.)

A. CEBOLLA CREEK, COLO., ORE. (× 60.)



B. RIVER CHALOUPE, CANADA, ORE. (× 60.)

pointing in others. No rule of general application can be formulated as to the behavior of the ores. Some deposits are very amenable to magnetic separation, and yield concentrates that require the admixture of only a small proportion of nontitaniferous ores to make a satisfactory ore mixture. Other deposits are less amenable and yield concentrates that would have to be mixed with three or four parts of nontitaniferous ores for furnace use. There are many other deposits in which the percentage of titanium separable in this way is extremely small, so that the utilization of such ores seems to depend on the discovery of a process that will make their use feasible.

The cause of the difference in behavior of the ores toward magnetic separation is revealed in their microstructure which is readily studied by the metallographic method. This method of investigation shows that the titanium occurs in these ores in the form of ilmenite grains in about the same order of magnitude as the magnetite grains, as ilmenite inclusions and intergrowths of microscopic size in the magnetite, and as an integral part of the magnetite molecule itself. The titanium occurring in the first form (as ilmenite grains) is readily separable by means of magnetic concentration, except in the finest grained ores with which the degree of crushing required would make such concentration impracticable. The titanium occurring in the two latter forms is inseparable by mechanical means. The percentages of titanium occurring in these different forms varies greatly in the different ores, and, consequently, every case must be tested for itself. A metallographic study of the ores of any deposit will at once decide the question of the amenability of the deposit to magnetic separation.

As regards chemical composition, except for their titanium content, the ores are very desirable. The coarser-grained ores are usually high grade in their natural condition, whereas a magnetic separation of the leaner ores yields a high-grade concentrate, with the deleterious constituents at a minimum.

As the iron industry at present demands large deposits of definite extent, the outlook for most of the deposits of titaniferous iron ore in the United States is not promising. As a rule the deposits are relatively small and of irregular extent and distribution. Further, they are lean to medium grade, and inaccessibly situated as regards transportation facilities. In other words, to put the deposits on a producing basis would require a heavy initial outlay of capital, which the size and irregularity of occurrence does not warrant. There are, however, the two large, high-grade deposits of Sanford Hill, N. Y., and Iron Mountain, Wyo., which are so readily workable that, despite their titaniferous character, their utilization within a few years seems certain.

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Bulletin 65 Petroleum Technology 7

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

OIL AND GAS WELLS THROUGH WORKABLE COAL BEDS

PAPERS AND DISCUSSIONS

BY

GEORGE S. RICE, O. P. HOOD
AND OTHERS

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PREFACE.

During the past few years the Bureau of Mines, in its study of problems relating to safety in coal-mining operations, has been considering the increased risks of mine explosions and the waste of coal that result from the present system or lack of system in drilling oil and gas wells in the coal fields. From time to time requests have been received from mine inspectors and other State officials that the engineers of the bureau investigate this subject and recommend such changes in common practice and such changes in State legislation as will help remedy both the increasing hazard and the waste of coal. Meanwhile, others have been studying the same problems, and at a meeting held in January at New Haven, Conn., the Association of State Geologists appointed a committee of its members to cooperate with the Bureau of Mines in developing remedies for this situation.

In this investigation, as in other of its investigations of mining conditions, the Bureau of Mines has adopted the practice of conferring with representatives of all parties at interest, with a view to procuring information concerning all phases of the subject, and also with a view to being able to develop and recommend for general adoption such changes in practice as well as in legislation as might prove to be effective by being both reasonable and enforceable.

In carrying out such a policy with reference to the subject under consideration, the Bureau of Mines invited the State geologists, mine inspectors, and a number of coal operators and oil and gas well drillers from the States interested to meet representatives of the Bureau of Mines in Pittsburgh, Pa., for a discussion of the subject during Februray 7 and 8, 1913. As a result of this conference, a committee representing all the interests concerned was appointed to consider all phases of the question, and to report its recommendations back to an adjourned meeting of the conference. This adjourned meeting of the conference will be held at an early date and its report will be printed in a later edition of this bulletin, with such additional suggestions as may be received by the Bureau of Mines. The bureau will be glad, therefore, to receive at the earliest practicable date any additional suggestions or any additional information throwing light on the matters discussed in this bulletin.

It is clearly understood that the inspection and control of all such operations as mining and well drilling lie within the province of the State. The work and the plans of the Federal Bureau of Mines are properly limited to inquiries and investigations and the publication of the results. Between State mine inspectors, State geologists, coal operators, well drillers, etc., in the different States, the bureau acts in these matters as a medium for the exchange of ideas, with a view to obtaining uniform legislation and methods in the different States.

JOSEPH A. HOLMES.

INTRODUCTORY STATEMENT.

By GEORGE S. RICE.

The need of protecting mines from the danger of inflow of natural gas from neighboring wells has become more apparent each year since it was found that oil and gas underlie the productive coal measures of Pennsylvania, West Virginia, Ohio, Indiana, Illinois, Kansas, Oklahoma, Colorado, and Wyoming, and, to some extent, the coal measures of other States. The proximity of the known gas and oil areas in the United States to the coal fields and the manner in which the fields overlap in certain States are shown in the map comprising the frontispiece (Pl. I).

As the result of many requests received from various sources a conference to consider the protective measures necessary to insure the safety of mining operations in gas and oil fields was called by the Director of the Bureau of Mines to assemble in Pittsburgh, Pa., February 7 and 8, 1913. To this conference were invited a considerable number of gas-well, oil-well, and coal-mine operators within convenient distance from Pittsburgh, as well as State geologists and State mine inspectors from a distance.

Gas and oil wells are also found near other kinds of mines, such as shale and limestone mines in western Pennsylvania, but these cases are so few that for this and other reasons it was deemed best to confine the present inquiry to the consideration of the questions involved in connection with coal mines.

It was also not thought advisable to consider the protection of mines from other kinds of bore holes than those for gas and oil, such as coal-prospecting holes, drilled either with churn or diamond drills, or salt wells and artesian wells, because little trouble has been experienced in mining operations from these classes of wells. Therefore the following papers and discussions concern almost entirely the following relations between coal mines and gas and oil wells:

(1) Protection of coal mines from gas and oil wells penetrating through: (*a*) Open workings; (*b*) inaccessible spaces or goaves; (*c*) pillars, or coal which may be left as pillars.

(2) The manner of drilling gas and oil wells passing through workable coal beds so as to provide protection for future mining.

The preliminary inquiries of the engineers of the bureau had disclosed that there was no uniformity in the methods of protecting mines against the leakage of gas from wells, that thousands of wells in coal fields are abandoned yearly without adequate plugging, and without surveys or records being kept to show the location of the holes, from which the casings are generally pulled. According to the State geologist of Pennsylvania, as many as 3,000 wells are being drilled annually in Pennsylvania, 2,000 of these holes being abandoned within a single year. It is such unplugged, uncharted wells that are the greatest menace in mining. The quantity of gas that the wells produce, although not sufficient to be commercially avail-

able, is enough, if it were to leak into a mine and be ignited, to cause explosions or fires, with possible loss of life.

The matters mentioned were fully considered in the papers read at the conference and embodied in this report. The papers outline the preliminary inquiries and examinations of the engineers of the bureau, and much of the discussion thereon by the members of the conference is considered worthy of inclusion.

In order that definite action on the protection proposed might be possible, an outline of suggested legislation was formulated by the engineers of the bureau and presented to the conference. It is embodied in one of the papers comprising this report. The outline should not in any way be considered as expressing either the conclusion of the conference or even of the engineers who presented it, but was intended merely to suggest topics for systematic discussion of the problems involved.

As will be seen from the report of the proceedings, it was decided that in order to obtain further suggestions for legislation along practicable lines, with a view to obtaining a maximum amount of protection for the minimum amount of expense, further consideration of the various questions involved should be given. The bureau was asked to submit a full account of the proceedings to those present and also to those unable to attend the conference. This publication is prepared in order that the data obtained may be distributed among those interested in this subject, either for action or for further consideration.

As will be noted from the report of the proceedings, the conference before adjournment planned the appointment of a committee consisting of the officers of the conference ex officio and three representatives from each of the following bodies: Gas-well operators, oil-well operators, coal-mine operators, State geologists, State mine inspectors, and officials of the Bureau of Mines. Each of these bodies was requested to appoint on this committee representatives who, so far as possible, would fully represent the industry in all parts of the country. In the event of failure to fill vacancies, the chairman (Mr. W. E. Fohl) was empowered to appoint persons to fill these vacancies. A temporary committee of one member from each body was appointed to assist the chairman in arranging for the general committee, which is to assemble in Pittsburgh on February 26, 1913. This committee is to give consideration to all the proposals made at the meeting herein reported, and to such as may be submitted by correspondence, and is to draft a tentative outline suitable for the groundwork of future legislation in each of the various States concerned.

The reader of these proceedings must bear in mind the difficulty in getting an exact report of what was said by each speaker. Frequently the discussion was of a conversational nature and at times was illustrated by chalk sketches on a blackboard, so that correct interpretation by the reporter and editor was often impossible, requiring the omission of some of the statements. Owing to the urgency of getting out the proceedings, it was not possible to submit the manuscript to the respective speakers. It is expected that reports of omissions of an important character and new matter will be submitted to the committee either by those present at the conference or by others. Therefore anyone who may desire to do so is invited to contribute suggestions bearing on the subject, which may be sent to O. P. Hood, secretary, care Bureau of Mines, Pittsburgh, Pa.

OIL AND GAS WELLS THROUGH WORKABLE COAL BEDS; PAPERS AND DISCUSSIONS.

By GEORGE S. RICE, O. P. HOOD, AND OTHERS.

OPENING OF CONFERENCE.

The sessions of the conference were held in the assembly room of the Engineers Society of Western Pennsylvania. At the opening session in the morning of February 7, Mr. Rice, on behalf of the Director of the Bureau of Mines, who was unavoidably detained in Washington, welcomed the members, briefly mentioned the suggestions received, especially those from a committee of the State Geologists' Association of America, that were evidently the outcome of a general desire for more satisfactory and uniform legislation concerning the drilling of wells and the reduction of the risks from mine explosions caused by the escape of the gas into the mine workings.

Those who registered at the conference were as follows:

Irving C. Allen, chemist, Bureau of Mines, Pittsburgh, Pa.
C. W. Baker, secretary and treasurer, Greensboro Gas Co., Pittsburgh, Pa.
L. F. Barger, general superintendent, Peoples Natural Gas Co., Pittsburgh, Pa.
A. C. Beeson, chief engineer, Pittsburgh-Buffalo Coal Co., Pittsburgh, Pa.
John W. Bolleau, Pittsburgh, Pa.
A. P. Cameron, general superintendent, Westmoreland Coal Co., Irwin, Pa.
B. H. Canon, United Coal Co., Pittsburgh, Pa.
John B. Corrin, assistant manager, Hope Natural Gas Co., Pittsburgh, Pa.
E. E. Crocker, vice president and general manager, South Penn Oil Co., Pittsburgh, Pa.
J. W. Devison, superintendent, Federal Coal & Coke Co., Grant Town, Pa.
F. W. De Wolf, director, State geological survey, Urbana, Ill.
W. C. Edwards, Parker & Edwards Oil Co., Pittsburgh, Pa.
W. E. Fohl, mining engineer, Pittsburgh, Pa.
John Gates, attorney, Philadelphia Co., Pittsburgh, Pa.
R. D. Hall, associate editor, "Coal Age," New York, N. Y.
A. G. Heggem, petroleum engineer, Bureau of Mines, Pittsburgh, Pa.
R. R. Hice, State geologist, Beaver, Pa.
O. P. Hood, chief mechanical engineer, Bureau of Mines, Pittsburgh, Pa.
L. M. Jones, engineer, Bureau of Mines, Pittsburgh, Pa.
G. L. King, division superintendent, South Penn Oil Co., Cameron, W. Va.

M. B. Layton, assistant manager, Manufacturers Light & Heat Co., Pittsburgh, Pa.

E. D. Leland, superintendent, Philadelphia Co., Pittsburgh, Pa.

W. L. McCloy, general superintendent, Philadelphia Co., Pittsburgh, Pa.

E. B. Moore, chief engineer, Consolidation Coal Co., Fairmont, W. Va.

A. J. Moorshead, president and general manager, Madison Coal Corporation, representing Illinois Coal Operators Association, St. Louis, Mo.

T. A. Neill, division superintendent, South Penn Oil Co., Mannington, W. Va.

W. C. Neill, attorney, Manufacturers Light & Heat Co., Pittsburgh, Pa.

John Rees, mining engineer, Youghiogheny & Ohio Coal Co., Pittsburgh, Pa.

George S. Rice, chief mining engineer, Bureau of Mines, Pittsburgh, Pa.

George W. Schluederberg, general manager, Pittsburgh Coal Co., Pittsburgh, Pa.

E. J. Taylor, chief engineer, Pittsburgh Coal Co., Pittsburgh, Pa.

C. B. Turner, superintendent, South Penn Oil Co., Mannington, W. Va.

E. A. Watters, chief mining engineer, Hicks Coal Co., Freeport, Pa.

W. A. Weldin, assistant chief engineer, Pittsburgh-Buffalo Coal Co., Pittsburgh, Pa.

David Young, State mine inspector, Freeport, Pa.

A number of other persons who arrived after the beginning of the conference did not register.

Mr. William E. Fohl, of Pittsburgh, was elected chairman, and O. P. Hood, of the Bureau of Mines, secretary. The formal organization of the conference having been completed, the first paper on the program, a general introduction to the topic to be discussed, was read by Mr. Rice.

GAS AND OIL WELLS IN COAL FIELDS.

By GEORGE S. RICE.

INTRODUCTION.

The problem of gas or oil wells sunk through or near coal mines and through the future coal reserves has gradually become more and more serious, not only because of the danger to the miner but also because of the increasing loss of coal left as pillars about the wells, and this loss will increase in the future unless means are found to insure safety to the miner and permit the coal to be extracted.

A prominent engineer told the writer recently that he has been obliged to report unfavorably to his client upon the purchase of a certain tract of coal land which he was examining on account of the numerous oil and gas wells on the property, which would seriously interfere with mining operations. Diagrams and plans will be presented showing how very close together wells have been drilled.

DANGER TO MINERS FROM GAS WELLS.

Fortunately, thus far there has not been a large loss of life in mines through explosions or fires caused by leakages of gas from wells, but there are several cases on record in which lives have been lost, and the possibility of disaster has been present in a number of instances. One special case, which has been frequently mentioned, is that of the Middleton and Enterprise mines, near Fairmont, W. Va., where natural gas under high pressure leaked from a well and entered the two adjoining mines, leading to local explosions in each mine and to the death of three miners. Fortunately this inleakage of gas occurred at night when but few men were in the mines, and the company that owned the mines had taken excellent precautions to keep the coal dust wet. Consequently the explosions were of small extent. This case is particularly interesting because the well was surrounded by a 100-foot pillar, and there was a coal barrier 100 feet thick between the two mines. Therefore the gas entered the mines not through the coal but through the floor and along a line 2,300 feet in length, according to Mr. Tarleton, general superintendent of the company, who described the explosion in a paper that appears in the Transactions of the West Virginia Mining Institute for 1911.

Deep wells are constantly being drilled in the coal fields of this country and are very numerous in the coal basins of Pennsylvania,

West Virginia, Ohio, Indiana, Illinois, and Kansas, as is shown by the accompanying map of a part of Greene County, Pa. (fig. 2). In Pennsylvania and West Virginia, mines not infrequently strike abandoned, unmapped wells. Many of these wells have been found filled with gas, and in some instances the gas has been lighted. Recently a serious fire was caused in a coal mine in the vicinity of Pittsburgh through burning gas which had been ignited when an unknown well was struck in mining. Fortunately this fire was not attended by an explosion, but it led to the mine being shut down for a couple of weeks, and the bureau's engineers and rescue crew, equipped with breathing apparatus, had to be called upon to investigate behind the fire stoppings. In a number of cases where wells have been encountered there have been narrow escapes from explosions. It would therefore appear that the mining industry hitherto has been more lucky than farseeing in failing to take precautions to make abandoned wells secure and to record the situation of these wells, which, though not giving gas in commercial quantities, make gas enough to be a serious menace in mining.

NEED OF SUITABLE LEGISLATION.

The dangers that threaten have frequently been pointed out by Dr. I. C. White, of West Virginia, president of the Association of State Geologists. At a recent meeting of this association a committee was appointed to confer with Dr. Joseph A. Holmes, Director of the Bureau of Mines, on this subject. Director Holmes also received requests from other persons to take up the question with a view to formulating suggestions that might lead to uniform legislation regarding oil and gas wells in the various States in which gas or oil underlies the coal fields.

Hitherto, legislation on gas and oil wells in all the States, except Ohio and Indiana, has dealt with the subject only from the standpoint of protecting the wells from one another. The Ohio laws, and to a lesser extent the Indiana laws, take some cognizance of the danger to mining from the proximity of wells, but the laws of both States are considered inadequate in this respect.

The problem of formulating laws is not one that concerns alone the coal operator and the inspector, for it also concerns the gas and oil well operators. The latter must be given the opportunity to search for and obtain the gas and oil they hold under option or ownership. It is manifest that they wish to take reasonable precautions for safety, but do not wish to be put to unnecessary expense or to be prevented from drilling wells in coal fields; therefore some means must be found by which the wells can be drilled through or in the vicinity of the coal mines without creating dangerous conditions. Three parties are therefore vitally concerned;

the gas and oil well operator, the coal operator, and the miner, whose safety is looked after by the State mine inspector. The State geologists are interested because of the desirability of conserving oil and gas as well as coal. The United States Bureau of Mines is concerned both in the conservation of all mineral resources and in the safety of miners. Hence it is desirable to have good as well as uniform rules and regulations throughout the country.

PRELIMINARY CONFERENCES.

As a preliminary to this general meeting, it was thought advisable to confer with those interested in the questions involved; consequently a conference was held a few weeks ago with some of the coal operators who had had experience in dealing with gas and oil wells drilled through mines; then followed a conference with representatives of the gas-well interests; and more recently a conference was held with a number of State geologists, representing several of the leading coal-mining States in which gas and oil wells have become or are becoming important.

After these preliminary meetings, those members of the Bureau of Mines who had been charged by the director with this investigation prepared a tentative outline of rules and regulations that might serve as a basis for possible legislation on oil and gas wells through coal beds. This outline was formulated to harmonize as well as possible with the ideas presented in the several preliminary conferences by representatives of different interests involved, regard being had for the fact that it would be wise to have such rules and regulations of a general character suitable for a basis for possible legislation in each of the States—the rules to be amended or modified in detail to suit the particular conditions that prevail in any particular State. These proposed rules and regulations, and the interpretation prepared by other members of the Bureau of Mines, will be read at this meeting.

Several features that were discussed at considerable length in the preliminary conferences have not been incorporated in the rules and regulations outlined, as it was felt that their incorporation would not be wise, and, in fact, would have to be rather a matter of gradual development and, in certain cases, of private agreement between the parties at interest. These questions are discussed here.

SIZE OF COAL PILLARS THAT SHOULD SURROUND GAS AND OIL WELLS.

Investigations and inquiry show that the pillars that have been left around gas wells in Pennsylvania and West Virginia vary from 40 feet in diameter or 40 feet square to 200 feet in diameter or square, the well being at the center of the circle or square. Inquiry

has been made as to the reasons underlying the determination of the size of these pillars, but it was found that there was no scientific basis for such determination, and that the various sizes of pillars merely represented the opinions of the parties concerned or compromises between interested parties. The courts, in rendering decisions, seem to have determined arbitrarily the size of pillars that must be left in certain cases, the size specified in each specific case presumably representing the concensus of the testimony that was presented.

In the preliminary conferences it was unanimously conceded that pillars of coal of any reasonable size did not prevent leakage of gas into the mine when there were defects in the well casings, but that they served to support the overlying strata and thus prevent fracturing or breakage of the well casing anywhere from the coal seam upward to the surface.

PILLARS AS ROOF SUPPORTS.

Regarded from this point of view, the question of the proper size of pillar around a well is a problem somewhat similar to that which has arisen as to the proper size of pillar to be left surrounding a mine shaft. Inquiry by the bureau has not disclosed any uniform system of determining the size of a shaft pillar, but a number of empiric formulas have been used by mining engineers in the various mining countries. These formulas are usually based on the distance of the coal bed from the surface. The ratio of diameter of the pillar to depth from the surface in the formulas noted is from $\frac{1}{4}$ to 1 to $\frac{1}{2}$ to 1; that is, if the coal is 400 feet below the surface the pillar should be 100 feet to 200 feet thick.

A formula that is used for barrier pillars in the anthracite fields, by a number of prominent companies and is approved by the mine inspectors of eastern Pennsylvania, takes into account the thickness of the coal. The formula is as follows:

> The width of the barrier pillar is equal to the thickness of workings multiplied by 1 per cent of the depth below natural drainage level, plus the thickness of the workings multiplied by 5.

The drainage level presumably is at or near the surface. If a seam were 9 feet thick and 300 feet below drainage level then the thickness of the barrier pillar would be

$$(9\times3)+(9\times5)=72 \text{ feet.}$$

Although this formula seems chiefly for protection against inrushes of water, it is manifest that the effect of crushing is also involved, for if the pillar were crushed it would not furnish the necessary protection. "The Coal and Metal Miners' Pocket Book" offers this formula for shaft pillars deeper than 700 feet: $r=3\sqrt{D\times t}$, in which r equals the radius of the shaft pillar, D the depth of

shaft, and t the thickness of seam. For example, in a shaft 900 feet deep and the coal bed 8 feet thick, the radius of the pillar should be 255 feet, or its diameter should be 510 feet.

The necessity of protecting a shaft is doubtless greater than that of protecting a gas or oil well. The shaft is the means of egress, and it is not simply the protection of the shaft itself that must be considered, but also the support of the surrounding buildings, including the engine and boiler plant; hence there is greater need in having large pillars around shafts than around wells.

It is manifest that the varying character of the strata in different districts affects the determination of the size of mine pillars to be adopted for oil or gas wells—that is, on the assumption that it is necessary to prevent any movement of the ground surrounding the casing all the way to the surface. If the roof over the coal bed breaks easily the fractures will probably extend upward in more or less vertical planes. On the other hand, where the roof is strong sandstone or limestone and the fall occurs on only one side of the well first, which is usually the case, there is a possibility that there may be a considerable pull toward that side which may carry the break diagonally upward, so that the plane of fracture may intersect the well before reaching the surface, in which event there might be danger of rupturing the casing, or at least of swinging it out of line.

It was pointed out by Mr. S. A. Taylor in the first preliminary conference (that with the coal operators) that where the cleavage of the coal is very marked, as in the Pittsburgh bed, a pillar should not be square, but should be rectangular, in order to get equal strength—that is, the pillar should be longer across the faces, or in a direction parallel with the butts.

The Consolidation Coal Co. in the Fairmont district, W. Va., is considering the necessity of leaving 200-foot pillars around the wells. Figure 1 shows gas wells drilled in a certain area and a projected plan of working with 100-foot pillars surrounding the wells. There are also indicated 200-foot square pillars, which would require revision of projected plan of workings, and shows how serious this requirement may be when the wells are close together, either as regards the laying out of the mine or the loss of coal in the pillars. A map (fig. 2) is presented of a portion of Greene County showing oil and gas wells. This brings up the question whether or not it is necessary to retain the pillar permanently.

EXTRACTION OF COAL SURROUNDING A WELL.

In the preliminary conferences the writer suggested that it might be possible to dispense with a coal pillar around an oil or gas well, provided an artificial pillar was substituted. This proposal is hardly

so radical as it may seem at first. The coal surrounding shafts has been extracted without detriment in some of the longwall mines in Great Britain, and, the writer was informed, in one longwall mine in northern Illinois. Where this plan has been employed the section

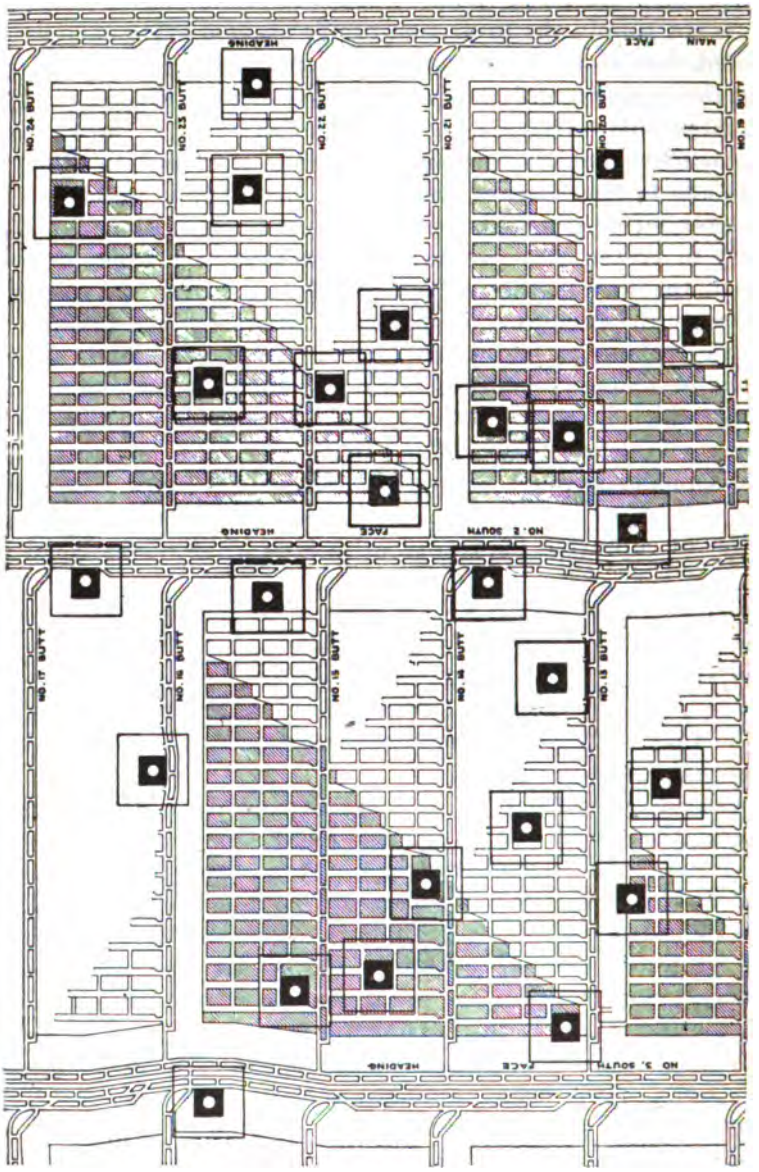

FIGURE 1.—Projected plan of mine workings with gas wells surrounded by pillars of coal.

from which the coal had been excavated has been carefully packed with broken rock and dirt, as is usually done in "longwall advancing," so that the movement downward is very gradual and regular, the strata all the way to the surface subsiding evenly and

without damage to the surface buildings, provided the longwall face advances regularly. The final surface subsidence, where the goave has been well packed with dry packing, is about 50 per cent of the thickness of the excavation—that is, in a bed of coal 6 feet thick the subsidence would be about 3 feet.

PROTECTION OF CASING WITH CLAY.

To suit the conditions of room and pillar mining, the writer suggests the following plan (see figs. 3 and 4): Lay out a room so that the drill hole will be approximately in its center. In drilling the well, make its diameter several inches larger than that of the outer casing, so as to leave a space of 1 to 2 or more inches surrounding

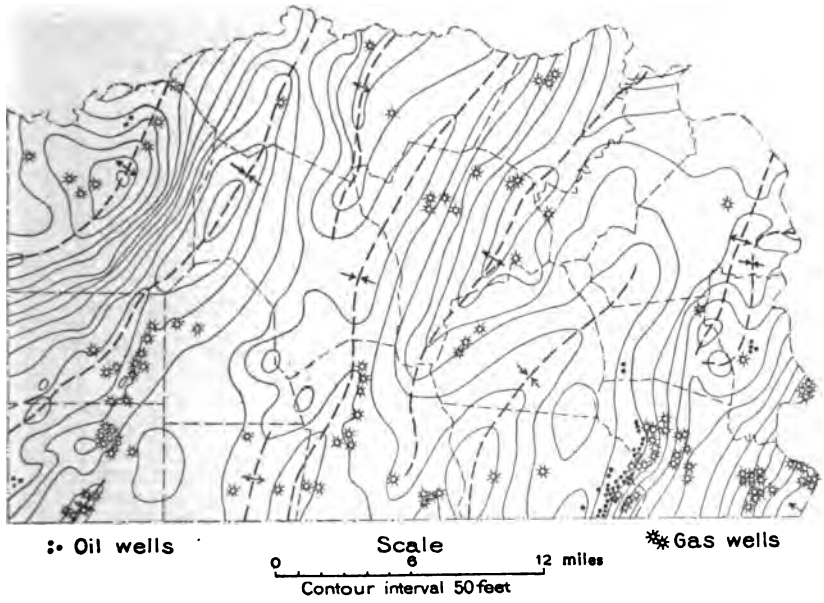

FIGURE 2.—Map of a part of Greene County, Pa., showing numerous wells, all of which penetrated coal beds. [From U. S. Geol. Survey Bull. 304.]

the casing. On the assumption that the outer casing is to go 30 feet below the coal bed, then the hole of the diameter stated is to be drilled an additional distance equal to the thickness of the seam, below where the shoe of the casing is to hang. The casing is then to be lowered and hung from a cross beam supported permanently at the surface. These cross beams should be steel channels or girders, 9 or 10 feet or more in length and strong enough to hold the weight of the casing. Before lowering the casing a gasket of rubber or canvas is to be fastened around it just above the shoe, so as to fit snugly against the rock or shale when the casing has been lowered to place and hung from the beam supports; a grouting of clay is then to be run in between the casing and the walls of the hole so as to fill completely this space from the gasket to the surface. The purpose of the

grouting is to allow the casing to slide freely, either at the bottom or the top, if the coal is excavated and the ground subsides. After the casing has been grouted, drilling and the putting down of the inner casing are to proceed in the usual manner. When subsidence of the ground occurs the arrangements spoken of will be equivalent to an expansion joint in a steam line. The inner casings and the gas tube, being rigid and fixed below, when the subsidence takes place will merely project that much farther above the surface of the ground, and will not be affected by the subsidence.

It is of course assumed that an opening will always be left at the top between the largest and the next inner casing to permit the

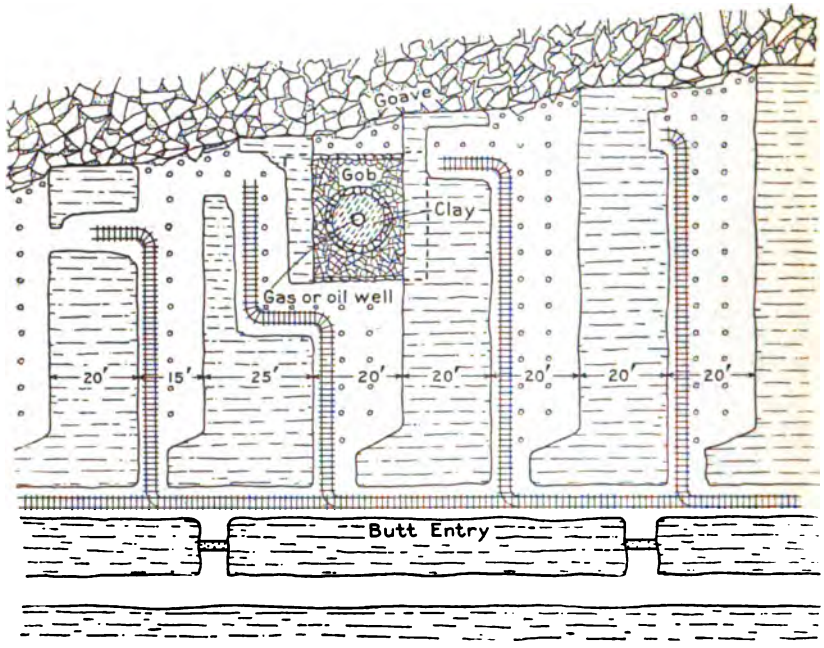

FIGURE 3.—Proposed method of protecting well where coal pillar has been removed. Plan.

escape of any gas that leaks into the space between the casings, in accordance with the plan employed in recent wells, where by agreement with the coal companies various precautions have been taken.

When the room mentioned above has reached the well, the casing should be protected from injury from cars or blasting by timbering; and if the roof is very poor, a timber cog should be carefully built around the well and packed tightly with clay. When the rooms in the vicinity are "up" their proper distance, before the pillars are withdrawn, clay should be brought from the surface (any ordinary clay will do), placed around the well casing, and tightly packed so as to make a cylinder from 6 to 8 feet in diameter. It will not be necessary to withdraw any of the timber to do this. The space sur-

rounding this clay cylinder back to the ribs should be filled with mine waste obtained from lifting the bottom or from fallen roof rock. When the ribs have been drawn back to a point opposite the artificial pillars, a thin curtain wall can be retained on either rib, merely to hold the gob in place. The purpose of the " gobbing " is to allow the roof to come down evenly, as the clay packed around the

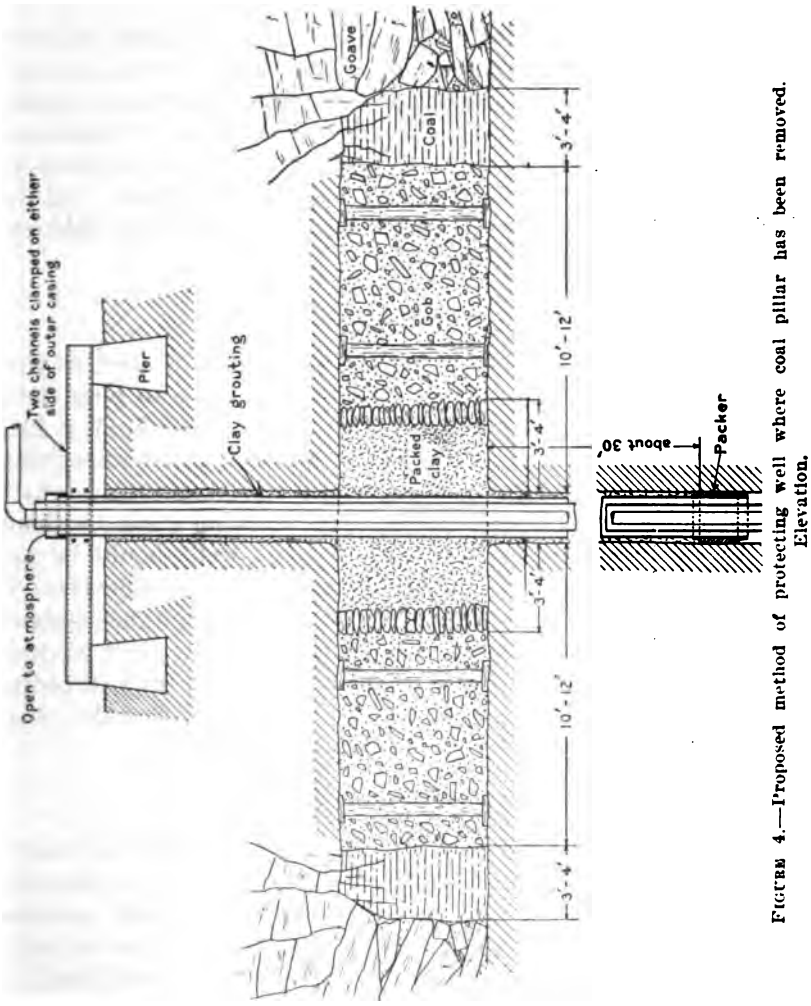

FIGURE 4.—Proposed method of protecting well where coal pillar has been removed. Elevation.

hole will prevent undue pressure at any point, and thus prevent distortion of the casing. Since above and below the coal bed the casing is surrounded with a layer of clay the roof may slip down without injury to the casing, or, if all the strata above come down and grip the casing, the latter will slide further down into the hole, which was previously drilled large enough to receive it. With such a plan properly carried out, the writer can see no serious

risk in not leaving a coal pillar, for even if a break of the outer casing occurs, which seems unlikely, there is at all times a vent to the surface between the inner and the outer casing.

LOWER COAL BEDS.

If abandoned wells are properly plugged from bottom to top, there would seem to be no need for other special precautions for deeper coal beds. It is manifest that in the great majority of cases these coal beds will not be worked for many years to come, perhaps 25, or 50, or as much as 100 years. It is manifest also that no casing would resist corrosion for such long periods, therefore it is believed that the project of restoring the strata to practically the same strength and impermeability that they had will be the best method. All the geologists in attendance at the recent meeting agreed that this was the best solution of this problem.

PLUGGING ABANDONED WELLS.

The geologists agreed further that the simplest and safest way of handling abandoned wells through coal mines will be to plug them tightly all the way from the bottom to the surface. (See fig. 7, p. 21.) The proposed details of plugging will be given in an accompanying paper. If holes are plugged so that there is no leakage of gas, there would seem to be no danger in laying out or working a mine without regard to an abandoned well, except that the well should be approached carefully, and tests should be made to see whether there is any sign of leakage of gas. Such lack of other precautions assumes that the plugging of abandoned wells in coal districts shall be done with an inspector present, preferably a gas and oil well inspector appointed by the State, and that a certificate showing that the plugging has been done properly has been duly recorded.

MAPS OF PROJECTED WORKINGS.

It is undoubtedly a most excellent plan for the gas and oil operator and the coal operator to come together and agree upon the suitable placing of holes. To do this intelligently the coal operator should furnish the gas and oil operator a plan of his projected mine workings. It is assumed that the site of the proposed well shall be carefully surveyed to and located, and that the map of the location and the record of the casing at least as deep as 30 feet below the coal bed that is being worked shall be matters of public record.

DRILLING GAS WELLS THROUGH INACCESSIBLE MINE WORKINGS.

The question has arisen in several cases as to the protection of a mine from gas leaking from a well drilled through an inaccessible opening or excavation, as, for example, where pillars have been

pulled. In the event of such leakage there would be danger of the goave filling with gas which would escape into the active workings. The writer does not attempt to discuss the relative merits of the two sides to the controversy, but as the courts have held that the parties owning the gas have the right to drill it is necessary to consider how drilling can be done with minimum danger. All have conceded the necessity of having at least one outer protective casing open at the

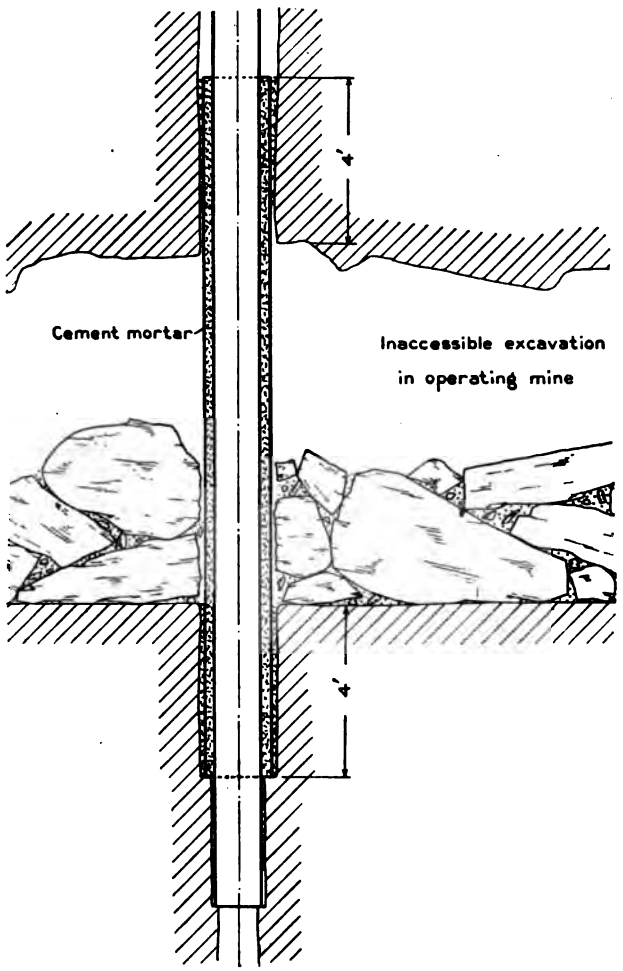

FIGURE 5.—Proposed method of protecting casing by use of cement mortar.

top to the atmosphere, but in the event that acid mine water may corrode it additional protection is necessary. One plan, proposed by Mr. McCloy, of the Philadelphia Co. (see fig. 5), is to make the hole large enough to insert an outer pipe opposite the coal bed and fill the space between it and the casing with cement. This plan seems to be excellent and worthy of trial.

An alternative scheme (fig. 6) offered by the writer might be used where the mine excavation is not too much filled with débris, and in some cases might be used to supplement the method previously described. The plan is as follows: After the drill has entered the excavated space and has passed through any loose rock to the bottom, fill the space with wet concrete composed of crushed rock or gravel, sand, and cement thoroughly mixed. This concrete is to be put in rapidly and will, if of proper consistency, form a conical heap with the apex at the top of the open space. A conical bit or tool fixed on the end of the string of tools should then be immediately lowered and used to spread out the top of the heap. More concrete should then be put in and should be spread out by the conical tool. After the concrete has been allowed to set, a hole can be drilled through it

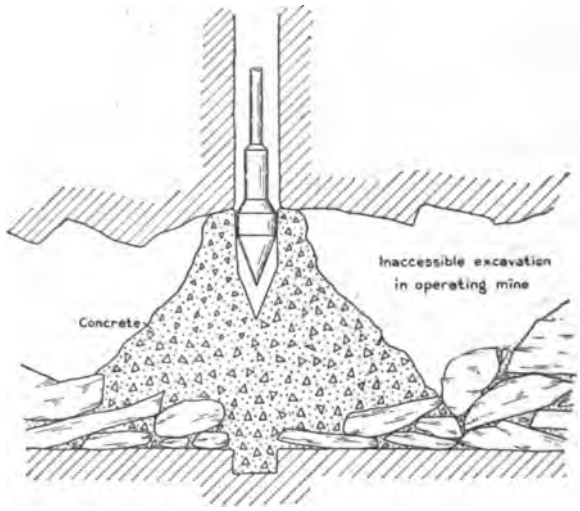

FIGURE 6.—Proposed method of protecting casing in inaccessible workings.

in the usual way. This hole may be a small one. Then liquid cement can be run in and forced under pressure to fill the interstices. After this has set it is anticipated that there will be a strong concrete column in the mine excavation. This column will be practically impervious to the mine water, and a hole of full size can be drilled through it as through solid rock and the casings inserted in the usual manner.

DISCUSSION.

A MEMBER. Referring to figure 5, do you fill the hole with concrete or with cement mortar?

Mr. RICE. It is filled with liquid cement and an inner casing to be lowered while it is still liquid. The surplus cement would then be removed from the inner casing.

Mr. McCLOY. The object is the protection of the inner casing from the mine water.

A MEMBER. [during the exhibition of fig. 7, relating to plugging an abandoned well]. Is this designed primarily to save the value of the coal, or to prevent the oil settling or otherwise moving into

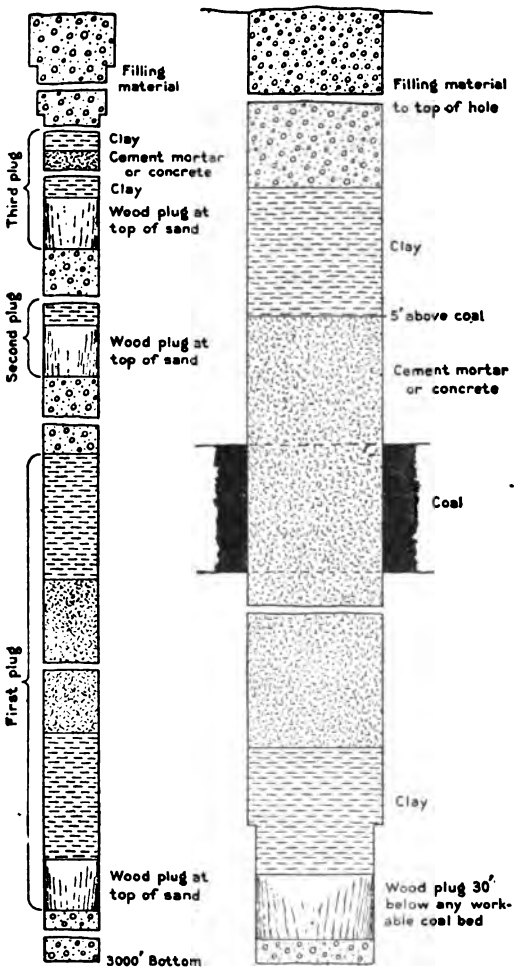

FIGURE 7.—Proposed method of plugging abandoned well.

other strata which might be detrimental to a large portion of the land?

Mr. RICE. It would lead to the same end in either case, I think, but primarily it is to avoid leaving a pillar. When the well is still producing oil or gas the plan presented in figures 3 and 4 might be employed.

A MEMBER. There is no doubt that the value of the coal recovered would fairly offset the cost of such a plan as this?

Mr. RICE. The cost of the coal recovered might not pay in the case of a single well, but I think it would pay where there are a large number of wells; so that it might be very difficult to mine the coal at all.

Mr. BOILEAU. Take the cheap coal, like some Illinois coal, $20 to $50 an acre, that would not be worth the saving, would it? I have seen an instance of gas coal where a man paid $200 for a 40-foot pillar around a well; that is, the man that drilled the well had to pay $200 for a 40-foot pillar around the well. I would rather have left the coal there in that case.

Mr. RICE. It is not a universal proposition in its application. I would like to see it taken under consideration. It might be found entirely impracticable.

Mr. WATTERS. I think that is a good scheme for a single well, but I do not think that it would be in all cases. For instance, suppose the well was drilled by a company in a new gas or oil field in which pretty rich wells are being struck, and suppose that the property line is 10 feet from a well, and an opposition company wanted to drill another well to get that gas, and they drilled a hole within 10 feet or 20 feet, how would that be protected?

Mr. RICE. That is rather too intricate to take up at this time.

A MEMBER. That will come up every day in good gas and oil territory.

Mr. RICE. My point of view is that if you protect with a pillar of coal that you have got to leave a big pillar, and if you are removing or pulling the room pillars the presence of a pillar around a well will interfere with the regular subsidence, which is essential in systematic pillar withdrawal.

Mr. FOHL. Then it is your view that the leaving of that comparatively large pillar would interfere with the subsidence of the strata.

Mr. RICE. That is what I was trying to bring out. In longwall work, if on account of property rights you have to leave a pillar, it usually causes more trouble to surface improvements than where you can take it out and let the settlement take place regularly.

Mr. MOORE (referring to figs. 3 and 4). That artificial pillar would be a sort of pedestal that would to some extent support the roof?

Mr. RICE. It would be an elastic support; it would gradually pack down to about one-half its original height.

Mr. MOORE. Suppose on that butt entry you have three wells, possibly not as close as the gentleman suggests, which you say are 300 feet apart; and if those three artificial supports tend to hold up the overlying strata to any great extent, wouldn't that be a source of a squeeze?

Mr. RICE. That would be a danger that would have to be taken care of as far as you could by making that support elastic. So it

settles sufficiently to allow the roof to break at the edge of the pillars being withdrawn, I think a squeeze would be avoided. To make it very elastic you would allow the clay and other material to be squeezed out laterally and perhaps omit some of the curtain pillars. The proportions of these as shown in figures 3 and 4 are not to be taken literally. It is just a suggestion of what may be done.

Mr. HALL. Hadn't that pipe better be held by something that would move in the hole instead of by something that is in the roof? The material which is to keep the clay in place, moving in the hole, instead of being underneath the roof?

Mr. RICE. We are providing that the hole below the casing be drilled large enough so that the casing can slide either down or up.

Mr. HALL. If the clay layer were thinner, you could put another pipe on the outside, and let the casing run up and down in that, and so need nothing more to protect it inside of the mine.

Mr. RICE. The object of this scheme is to provide a cushion for the roof, to prevent it swinging to one side and rupturing the casing. It is to make the strata in the vicinity descend evenly when the adjacent coal pillars are pulled, and I think that an outer casing suggested by Mr. Hall will be unnecessary if the holes are drilled somewhat larger than they are ordinarily, and are filled outside the casing with clay grouting.

Mr. FOHL. Have you stated the approximate dimensions that you would have for this artificial pillar?

Mr. RICE. I have in mind that the clay pack surrounding the hole would be about 8 or 10 feet in diameter, and outside of that would be built up a gob wall. You would pack the clay as the wall was built, and perhaps wet it; much depends upon the circumstances in each case. From experience in northern Illinois, where you pack with rough shale material, the subsidence is about 50 per cent, and that is obtained in a year or two. Most of the settlement underground takes place in the first two or three months; but the surface is affeced more slowly, depending on the depth.

Mr. HALL. Is it the idea that there is another butt entry parallel with that?

Mr. RICE. Yes. It is intended to represent the ordinary conditions in the Pittsburgh field. The dimensions (fig. 3) may not be quite right as to width of rooms and pillars, that is immaterial; the scheme is simply presented as a general plan.

Mr. FOHL. We are certainly greatly indebted to Mr. Rice for the time he has spent in the preparation of this subject, and we hope that a little later these illustrations and this paper will receive a full discussion. We will proceed with the next paper on the program: "Mine Troubles Due to Proximity of Gas and Oil Wells," by Mr. L. M. Jones.

MINE TROUBLES DUE TO PROXIMITY OF GAS AND OIL WELLS.

By L. M. Jones.

INTRODUCTION.

The relation of gas and oil wells to nearby coal mines has been investigated by inquiring into the past and the present practice of operating each and by noting wherein accidents and troubles have resulted or are likely to result. Published references to several sources of trouble have been found, and as many as time would permit have been investigated in the field with a view to determining causes and remedies. The cases on which information has been obtained have been classified according to the cause of the accident or trouble and are discussed below.

ABANDONED WELLS UNCHARTED AND IMPROPERLY PLUGGED.

Until very recently there has been no legislation in any of the States requiring the filing of a statement giving the exact location of a drill hole, and the legislation covering plugging has for the most part not covered the matter thoroughly, and has been difficult to enforce.

Consequently in some of the old oil and gas fields many of the wells drilled have been uncharted and have been ineffectually plugged or not plugged at all. Where these wells have passed through coal beds that at a later period have been opened, several accidents have been caused by the unexpected piercing of the wells and the resulting liberation of large volumes of natural gas.

MINE FIRE CAUSED BY PIERCING AN UNCHARTED ABANDONED WELL.

A serious mine fire was caused at a mine at Vandergrift, Pa., on July 30, 1912, by the lighting of gas that entered the mine after the striking of an abandoned gas well.

A machine runner and a scraper were undercutting the coal with a chain machine at the right corner of the face in a room about 11 o'clock in the morning. The machine cut into the uncased well hole and a large volume of gas immediately entered the mine. The gas was lighted by the open lights of the miners and they were forced to run from the working place. As they ran out of the room the flaming gas followed just beneath the roof. Several unsuccessful attempts were made by the foreman and others to reach the face to get the mining machine.

At 4 p. m. brick stoppings were started at the mouth of entries 19 and 20. They were completed that evening.

Later by digging on the surface the top of the well hole was found. It was cleaned out to a depth of 715 feet (500 feet below the coal), where there seemed to be an obstruction. The hole was then cased in the following manner:

An 8-inch pipe was inserted to a distance of 4 feet below the surface; then 4-inch pipe was inserted to a point 8 feet above the coal and at its end a 3⅜-foot by 7⅝-inch sleeve wall packer was placed. Inside the 4-inch pipe a 2-inch pipe was inserted to a point 150 feet below the coal, and a disk wall packer was placed at its end. This arrangement allowed a free vent from below the coal to the surface.

After an investigation with helmets inside the sealed area had indicated that the fire was out the entries were reopened.

SEEPING OF OIL INTO MINES.

In a number of coal mines oil has oozed from the coal from wells whose position was not accurately known, the surface indications having entirely disappeared.

In one mine in the Pittsburgh district, after two rooms off a certain butt entry had passed through a clay seam, oil oozed from the coal face. Work in the rooms was immediately discontinued, and samples of air were taken at the face to determine whether any natural gas was leaking into the mine. The samples failed to show the presence of any natural gas, although three abandoned wells were near the point of sampling. Two of these were within 250 feet from the points where the oil appeared, and the third was about 400 feet distant. All three wells were supposed to have been plugged before abandonment.

There have been other cases where oil from a neighboring well has entered a mine through the coal. In one mine the quantity so entering necessitated the installation of a pump for its removal.

WELLS WITHOUT VENT FOR CASING GAS.

A second group of troubles arises from wells that have been improperly cased and sealed, some of which have no vent to the surface for gas collected within the outside casing.

MINE EXPLOSIONS CAUSED BY NATURAL GAS FROM WELLS.

EXPLOSIONS IN CONSOLIDATION COAL CO.'S MINES NOS. 47 AND 49.

Probably the most serious accident that has yet occurred from natural gas entering a mine resulted from an improperly capped well that had been drilled through mine No. 47 of the Consolidation Coal Co. in West Virginia. This accident occurred December 19, 1910, and was described by C. H. Tarleton before the West Virginia Min-

ing Institute in 1911. The well in question was so small a producer that it had been closed in, as its commercial value was questionable. It passed through the mine at one end of a coal pillar 125 feet wide and 380 feet long. In some manner not definitely known the gas from the tubing leaked into the casing, which was closed at the surface. The high pressure, said to be possibly 1,000 pounds to the square inch, forced the gas through the rock strata underlying the coal, and the gas entered mines Nos. 47 and 49 at a number of points in practically a straight line extending southwest parallel to the coal contours of the Pittsburgh seam for a distance of 2,300 feet from the well. As these mines were nongaseous, open lights were used; the gas in both mines was lit by the open lights, explosions resulting. Fortunately the coal dust in both mines was damp, so that the explosions were not so widespread nor so violent as they might otherwise have been.

In mine No. 47 the explosion occurred at 6.50 a. m., while the men were going to work. Three of them lost their lives owing to inhalation of flames; the others succeeding in getting outside without being caught in the afterdamp.

In mine No. 49 gas blowers in rooms 10 and 11, off the third right butt entry, off the main south face, were lit about 8 o'clock p. m., December 18, by a pumpman, who was the only person in the mine. He succeeded in putting the fire out with his coat. At 5 o'clock the next morning he lit the gas in the third face entry and an explosion resulted. Fortunately the explosion ruptured an overcast near where the pumper was thrown, and the short-circuiting of the air prevented his suffocation by afterdamp.

The explosion in this mine caused considerable damage inby the point of ignition and started a mine fire in room 5, on the first left butt entry. Eight stoppings were required to seal off the fire area.

Samples of gas were taken in both mines, and the analyses indicated that natural gas was present, as a considerable percentage of ethane was found. When the well was opened the gas blowers in the mine began to diminish and eventually almost entirely disappeared.

PEORA EXPLOSION.

Another mine explosion, due to the lighting of inleaking natural gas, occurred at a country mine on the Mary Chalfont farm, at Peora, Harrison County, W. Va., November 22, 1912.

A man and two boys entered the pit, which has only a single entry, between 6 and 7 p. m. As they were going in they met a neighbor coming out with a wheelbarrow load of coal. It is reported that the neighbor had fired a shot just before he came out, but he did not say anything about it. When the three reached the face the man noticed that the roof was not in good shape, so he asked one of

the boys to raise the open light in order that he could see it better. As soon as the boy raised the light an explosion occurred. The man managed to get out, but the two boys were burned to death. The neighbors say that the flame extended 100 feet from the opening and was 40 feet in height. A wheelbarrow was thrown from inside the pit over 150 feet from the opening. Some timbers from the mine were also blown 150 feet from the opening. There was considerable gas coming from the ground around the outcrop and this took fire. Some person went to the gas well about 500 feet from the opening and opened the valve to the casing. The pressure was immediately relieved and the flame died down at the pit mouth and along the outcrop.

The explosion started a fire in the mine and it burned from November 22 to 24, when it was put out by Mr. Tarleton and some other men.

At some previous time the casing had been pulled out of the well, but the tubing had been left and the well remained in use. Only a little 6-inch casing had been left and this had been capped at the top and the valve had been closed, as had also the valve on the tubing. It is probable that the casing had been closed for some time and that the gas had broken into the mine as a result of the firing of the last shot. When Mr. Tarleton was there on November 24 there was still 2 feet of gas at the roof. By January 18, the pit mouth had been partly filled and water had backed into the mine so that it was impossible to get in very far. Two samples of gas were taken 10 feet inside the opening, but in the half light no gas cap could be seen. The well is supposed to be just 515 feet from the mine opening and the single entry of the mine is reported to have advanced to a point within a few feet of the well, but it is uncertain just how far the entry is in, since, so far as is known, no accurate measurements have been made.

REYNOLDSVILLE EXPLOSION.

Another instance of gas leaking into a country pit occurred near Reynoldsville, W. Va., on the Parkersburg branch of the Baltimore & Ohio Railroad. A man had gone to the face of the entry, 80 feet from the opening, and had lit a shot. He then went outside and waited until the charge had fired. The shot caused a violent explosion which blew two cars standing in front of the mine a considerable distance. After the explosion it was found that the mine contained large quantities of explosive gas. A well 1,000 feet from the opening was opened, but the gas still seemed to leak into the mine. A well 1,500 feet distant and in direct line with the entry was then opened and immediately the gas ceased coming into the mine.

CLARKSBURG EXPLOSION.

In the proceedings of the West Virginia Coal Mining Institute for 1911, Mr. Frank Parsons, district inspector in West Virginia, told of a mine near Clarksburg, W. Va., which had had an explosion, probably caused by the lighting of natural gas that had leaked in from a gas well. His reason for believing that the gas was natural gas is given below:

> The mine was in only 80 feet from the outcrop. A mine only 80 feet distant had been opened out completely and abandoned without ever detecting gas. Since abandonment of this mine four gas wells had been drilled in the neighborhood so that it was probable gas was leaking from one of these wells.

LEAKAGE OF GAS FROM WELL.

In another mine in southwestern Pennsylvania there is an abandoned well located in a pillar near the forks of two entries. Air samples taken two years ago in an air current from this section showed 0.87 per cent of natural gas, which was equivalent to 252 feet of natural gas a minute. Analyses made recently indicate that gas is still being given off but in decreasing volume.

The instances cited show that greater precautions should be taken to seal properly all wells from the coal beds and to provide a vent to the surface for any gas that may collect in the casings.

UNPROTECTED CASINGS.

There is always a possibility of danger from wells that pass through the workings of an operating mine if the well casings are unprotected. I have heard of no case where serious accident has resulted in the mines from such a condition, but many such wells are a constant menace to the safety of the mine on account of the fact that the exposed condition of the casings renders them more likely to be injured than if they were inclosed by a protecting wall or pillar.

BRIER HILL MINE EXPLOSION.

In the bituminous report of the Pennsylvania Department of Mines for the year 1895 James Blick, State mine inspector, reported an accident at Brier Hill mine, October 17, 1895, in which three men were badly burned, one of them dying 12 days after the accident.

The accident was due to the lighting of natural gas that had leaked into the mine from an oil well. The well had been drilled through one of the rooms about 2 years previous while the pillars were being drawn. The remaining pillars in the immediate vicinity had been left standing in order to support the strata and to protect the well. By reason of the fact that part of the coal had been taken out, the part remaining was not strong enough to carry the weight of the overlying strata, the result being that a creep had begun to overrun

that part of the mine. The continued subsidence of the strata finally broke the casing of the well, allowing the gas to escape. Three men were working only 300 feet away and the gas was lit by their open lights. Inspector Blick considered it very fortunate that the gas was lit so soon and so gave warning, as a short time later it had spread through a considerable part of the mine. There were 160 men in the mine at the time of the accident.

As the mine was ventilated by means of a furnace, the fire was immediately put out, so as to prevent ignition of the gas in the return.

An investigation indicated that the pump rods, tubing, and casing of the well were broken near the coal bed. The casing had been bent considerably out of line before being ruptured. The casing at the surface was connected to a gas line that supplied gas for firing some boilers. The inspector was uncertain whether the gas that entered the mine had come up the casing from the gas sands or had come down the casing from the pipe line, which was also connected to other wells.

BENDING OF CASING.

In a western Pennsylvania mine, where the casing of an oil well passed through a mine working, after the room pillars had been pulled, subsidence of the overlying rocks bent the casing to such a degree that the rods in the well could no longer be operated. In cases of this kind it would probably be impossible to plug the well effectively on abandonment, so that the well would remain a menace both to present mining and to future deep mining.

CASING BENT BY WRECKS.

In another mine near Pittsburgh the casing of an abandoned well imperfectly plugged is between the haulage tracks on a parting, and wrecks of trips have hammered it to such an extent that the casing midway between the bottom and the roof is 7 inches out of line. The inside of this casing is plugged with concrete for 50 to 60 feet below the mine floor, but some gas and salt water rise on the outside of the casing into the mine. The plugging of the well is not satisfactory, because, although little or no gas rises through the casing to the surface, the gas does come up outside the casing. The well should have been plugged below the casing and the casing sealed with cement.

PROTECTION OF EXPOSED CASINGS.

Several methods are used to protect casings exposed in mine workings. One large company has adopted the plan of building around the well a 13-inch wall of brick set in cement mortar.

Concrete walls have been built around many wells to protect them from injury.

If pillars are to be withdrawn, G. S. Rice, chief mining engineer of the Bureau of Mines, has suggested that a pack wall of slate be built around an interior filling of clay (fig. 4). This wall would partly support the roof and prevent the severe bending or rupturing of the casing by sudden falls of great weight.

DANGER FROM DRILLING OPERATIONS.

The manner of drilling certain oil wells has been a source of annoyance and of disagreement between mining and well-drilling companies. The drilling has been so conducted as to inconvenience the operation of the mine and even to endanger the safety of the miners and mules.

In some especially aggravating cases the well has been drilled irrespective of whether there were mine workings beneath or not.

MULES KILLED.

I have heard of no case in which a human being has been injured as a result of inconsiderate well drilling, but I know of two instances where mules were killed.

One of these occurred in Illinois 15 or 20 years ago when a string of tools in an oil well over a mine broke through into an underground stable and killed a mule. More recently in Pennsylvania well-drilling tools broke into an entry and killed a mule.

LEAKAGE OF GAS AND OIL DURING DRILLING.

A well drilled through the Rice mine near Bergholz, Ohio, some time ago gave trouble. The drill passed through a room and the casing was driven down 10 feet below the coal. The hole was then drilled to a depth of 700 feet to the Injun sand, where oil and some gas was struck. The oil and gas came up outside the casing and, although a cement block had been built in the mine around the casing, entered the mine through crevices.

Mining and drilling were both suspended until changes could be made. A second string of casing was placed in the well with rubber packers above the oil-producing sand and 60 feet below the mine floor. Between the original and the new casing liquid cement was run from the packer to the surface. No further leaks were found in the mine.

INFLOW OF SURFACE WATER.

Another possible source of damage from drilling operations through mine workings is the inflow into the mine from the drill hole of large quantities of surface water. Such a case occurred in

an Ohio mine. The well had been cased to a point below the coal and a concrete block had been built around the casing, but the flow of water could not be stopped. The casing was pulled out and the hole filled with cement to a point above the roof. The casing was then lowered into the liquid cement and the cement allowed to set inside and outside the casing. Afterwards the hole was drilled through the cement inside the casing. This method stopped the inflow of surface water.

DRILLING THROUGH ABANDONED WORKINGS.

It has sometimes been difficult for the gas or oil company and the coal company to agree as to the safety of drilling through abandoned mine workings.

In the case of the Monongahela River Consolidated Coal & Coke Co. against the Greensboro Gas Co. the coal company desired that the drilling of a well through abandoned workings of the Snow Hill mine be enjoined, contending that such a well would be a danger to the Snow Hill mine and to neighboring mines connected with it. As the well passed through inaccessible workings no protecting barrier could be built around it, and consequently it was liable to be injured by the subsidence of overlying strata or by the corrosive action of acid mine water on the casing.

A temporary injunction was later dissolved on the ground that the gas company had a legal right to drill the well through the coal stratum. The gas company was instructed, however, to fill with liquid cement the space between the 10-inch casing and the walls of the well and the space between the 8-inch and 10-inch casings.

DESIRABILITY OF AGREEMENT.

The instances cited show the desirability of an agreement between a well-drilling company and a mining company as to the location of wells through mine workings, also the desirability of an agreed method by which such work shall be done so as to safeguard the mine from inflows of gas, oil, or water and so as to safeguard the wells from damage due to mining operations.

FIELDS HAVING MANY WELLS.

In some fields the wells are very close together, being only 100 to 400 feet apart, and in such territory mining development is very difficult.

In the Canonsburg, Pa., field and in the Scio, Ohio, field some of the wells are very close together. A photograph of a district in Greene County, Pa., also shows there are many wells. Mr. I. C. White, at a West Virginia coal mining institute meeting in

1910, stated that similar situations exist in Wetzel County, W. Va., where some farms had 200 to 300 wells; about one-fourth of these had been abandoned without being charted.

LOCATION OF WELLS WITH RELATION TO PROJECTED MINE DEVELOPMENT.

A plan (fig. 1) showing a projected mine development in West Virginia gives an illustration of the seriousness of the problem when coal pillars must be left to protect wells in the territory to be developed. In this case the projected underground development work was staked out on the surface and the wells were located so that 100-foot pillars surrounding them would not necessitate curving the entries. When the mine development is actually undertaken, it may be found advisable to increase the size of the pillars to 200 feet so that the entries will have to be driven differently from the manner indicated.

In the case last cited the drilling company was willing so to locate its wells as to inconvenience the coal-development work as little as possible. Most of the large coal and gas and oil companies in Pennsylvania and West Virginia have been willing to cooperate in this manner so that the development of either would inconvenience the other as little as possible.

When a gas or oil company desires to drill a well in a certain place it furnishes the coal company with the proposed location to determine whether there is any objection on the part of the mining company. Where wells would interfere with projected development work or would be difficult to protect, changes in location are made to meet these objections.

After Mr. Jones had read his paper, on motion, the conference adjourned from 12 o'clock until 1.30 o'clock p. m.

The afternoon session was called to order at 2 o'clock by Chairman Fohl, who called for the next number on the program, "Suggestions for Legal Regulations of Bore Holes Passing through Workable Beds of Coal," by Messrs. O. P. Hood and A. G. Heggem.

Mr. Hood stated that before starting to read the paper he wished the members to understand that it was the opinion of the Bureau of Mines that these suggestions should be left in the hands of the members. The whole object of the conference, he said, was to get the help and suggestions that are so useful and are so much needed in every way by the bureau in its endeavor to collect information. It was for the members, he said, to discuss this information, and, to use a rough phrase, "to rip it up the back" if they saw fit, and there should be no hesitancy on the part of anyone to give all the criticisms that he might feel moved to give.

Mr. Hood then read the paper.

SUGGESTIONS FOR LEGAL REGULATION OF BORE HOLES PASSING THROUGH WORKABLE BEDS OF COAL.

By O. P. Hood and A. S. Heggem.

In the following suggestions for regulations in the matter of bore holes passing through workable beds of coal as herein presented, an attempt is made to harmonize as far as possible the information and advice that have so far been available to the bureau. Suggestions have come from those interested in the safety of miners, those interested in mining operations and in the boring of gas and oil wells, and those interested in geology and the conservation of the coal, gas, oil, and other mineral resources of the country. In the suggested regulations the authors have not attempted to adopt legal phraseology nor to offer an entirely complete set of rules. The proposed regulations are expected to form a basis for discussion and further suggestion by this conference, to the end that suitable laws may be suggested to the various States in order to make mining safer and to prevent mineral waste.

It seemed to be the general opinion that some form of inspection is necessary in order to make operative any laws to be enacted that require specific methods and results. It is therefore suggested that there be a chief inspector of gas and oil wells; that he have sufficient help of a permanent and temporary character to meet the exigencies of the well-drilling business; and that these officers have certain duties prescribed.

In order that it shall be possible to locate accurately a well that goes through a workable coal bed and may at some time be within a mine area, it seems desirable to require a license to drill, this license depending upon the filing of proper maps and records. In case a location is proposed that may be detrimental to a coal mine, either from the standpoint of safety to the miner or that of the economical working of the mine, provision is made for a conference between the three parties interested in such location, these parties being the mine owner, the well owner, and the miner as represented by the State mine inspector.

The inspector of gas and oil wells is given power to change the proposed location within reasonable limits to the end of insuring a safe and equitable condition. It is also required that the inspector shall keep records in his office so that at any subsequent time it will be possible for a mine operator to locate accurately abandoned wells and not have to proceed in ignorance of their existence.

In order that the wells shall be properly plugged when they are abandoned it has been suggested that the well driller shall furnish a bond, which is to be returned to him when the plugging has been properly done. The inspector is required to supervise this plugging, so that a record becomes available of the method employed and the satisfactory accomplishment of the operation.

In making a provision that wells that have suspended operation shall be maintained in a safe condition by the well owner, it is believed that formal abandonment rather than neglect will naturally follow.

The suggested regulations cover only those wells that are put down through workable coal beds. A definition of "workable coal bed" is so difficult that it has not been attempted, and the responsibility for a reasonable interpretation of this term is placed on the State geologist whenever the matter becomes one of dispute. It is believed that some such official would be expected to interpret this term in view of the intention of this act, namely, to protect the miner while working beds of coal that may be used during the reasonable life of any well. It is not the intent to require special protection for a seam of coal that is physically workable and yet whose exploitation is probably so remote as to make it more than probable that the well will have been abandoned and properly plugged before there is need of the bed.

The location of a bore hole should be so accurately determined and recorded that the well can at any time be relocated, even after all surface indications have disappeared. For this purpose it is necessary to have the survey made by competent persons and to refer to established boundaries, which can be located only by reference to at least three monuments or reference points. It is also desirable that there should be uniformity in the matter of the scale of the maps submitted.

In order that gas may not be drawn into the ventilating system of any mine, a minimum distance from a mine opening has been provided, and a similar minimum distance to those buildings that are vital to the safety of operation of a mine, so that in case of fire at either wells or mine buildings the risk shall not be increased.

It seems evident that a bore hole should not go through any mine haulage-way or airway, and in order to prevent this occurrence, requirement has been made that the well shall be 15 feet from such mine haulage-way or airway. If pillars are considered necessary about such a bore hole it is believed that the supporting power of the haulage-ways ribs will be sufficient, as it seems to be generally admitted that a coal pillar surrounding a bore-hole casing can not be made to serve the purpose of keeping out gas that may leak around the casing.

In requiring casing from surface water, it is believed that after the casing that goes through the coal seam has been properly placed and packed as required, the casing used for excluding the surface water may be withdrawn. The double casing required through any coal

seam is deemed necessary in the event that a mine should be opened and the casing possibly exposed to corrosion or to injury from the movement of ground. In order that the ground pressures may be applied over a considerable surface of the casing, it is required that the double casing extend into the floor a sufficient distance; that it be surrounded with a clay pack; and that the second casing shall be brought into play through a similar clay pack. (See figs. 8 and 9.)

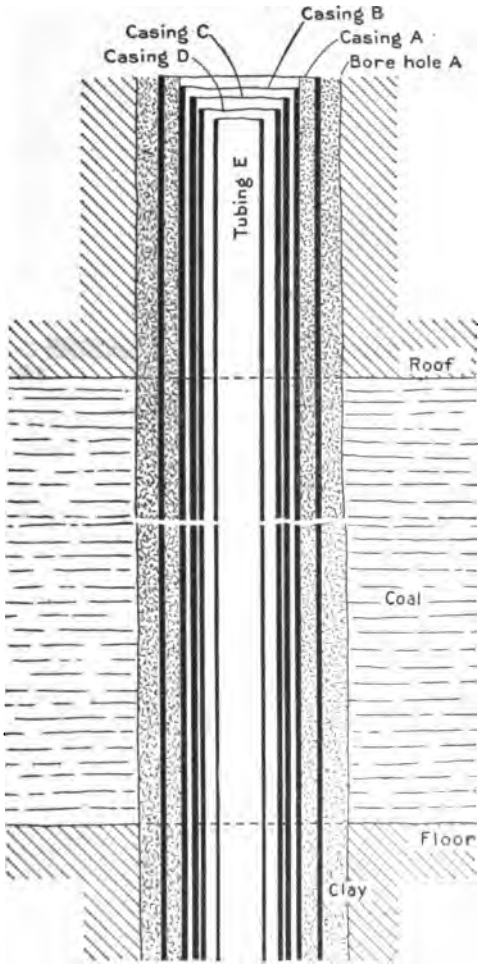

FIGURE 8.—Proposed method of sealing casing passing through solid coal.

The use of clay rather than cement for a packing material was preferred in order to avoid possible cracks through which corrosive water might enter if the outer casing was penetrated, and also to prevent the localizing of stresses on the second casing by a hard connection between the two casings. The packing material also serves to prevent leakage of gas into the casing space even if the casings are deformed by ground movement.

The second casing, which is required through any coal seam, may

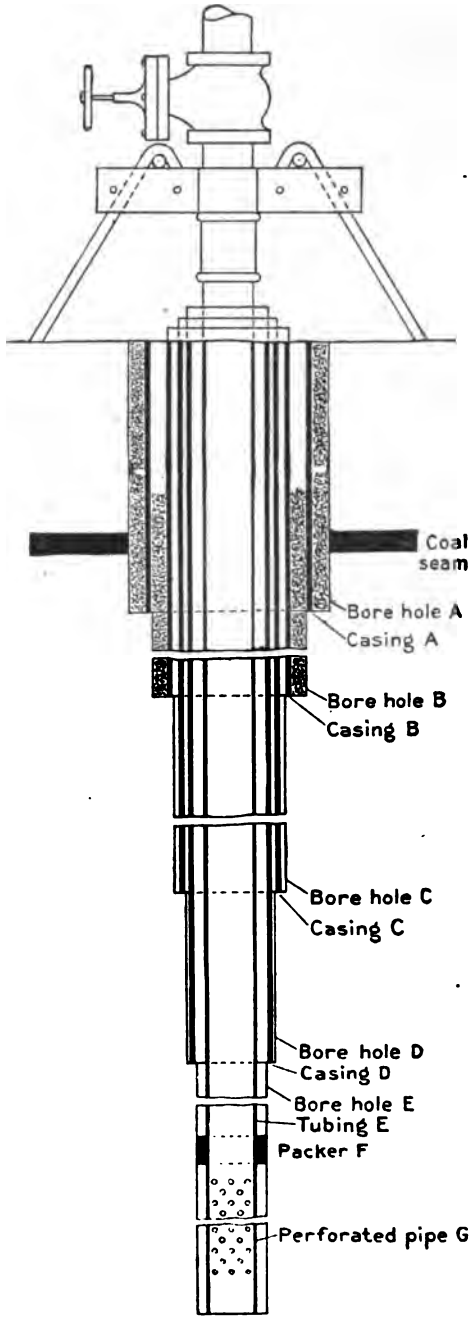

FIGURE 9.—Proposed method of protecting exposed casing.

extend downward to any depth beyond that prescribed, but in the

event that it is stopped 10 feet below the first casing, requirement is made that in cementing it the cement shall not extend up to the first or outer casing. The drawing of the outer casing when the well is abandoned is thus facilitated.

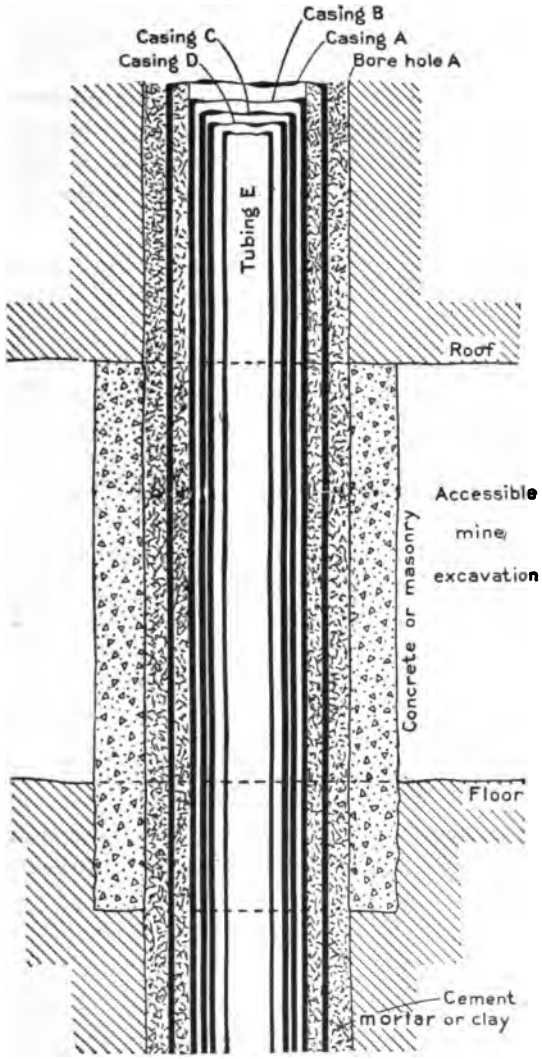

FIGURE 10.—Method of protecting exposed casing with concrete or masonry.

Where the casings pass through a mine excavation, a further mechanical protection, embracing a suitable wall and a clay pack surrounding the outer casing, is prescribed with the intent of preserving it from injury and corrosion. (See fig. 10.) Where such casting passes through an inaccessible mine excavation, attempt has been

made to provide a similar protection by prescribing a cement covering held within an outer metal tube acting as a form. (See fig. 7.)

In prescribing the arrangement of casings and casing head, endeavor has been made to insure a free vent to the atmosphere from a point in the well below any of the casing used and to insure that this vent shall be maintained by openings that can not be readily closed.

Where oil is allowed to rise within a casing above the floor of a mine, as may happen during disuse of the well, it may find its way past the several casing seats and into the mine, so that the requirement that the level of any such oil be kept below the mine floor seems necessary.

The abandoning of a well has been made a formal matter, so that a proper record shall be made and there shall be assurance that the work is properly done.

In prescribing a plugging method, attempt has been made to keep the method as simple and as general as possible. The method is based on the theory that although clay alone may make a good stopping, it may flow and not be maintained in its proper place. Although the cement mortar or concrete is apt to be porous, to deteriorate in oil, and to form a plug of doubtful tightness, it has the quality of staying where it is put. To combine the desired qualities the proposed regulations prescribe that there be a 25-foot cement plug with 5 feet of clay at each end, so that if there is any movement of fluids through the cement the clay will be carried into small openings and eventually close them. It also seems desirable that there be a hard, solid filling through any coal seam, so that if the well hole be struck by mining tools it will be immediately recognized.

The draft of the proposed regulations follows.

DRAFT OF PROPOSED REGULATIONS IN THE MATTER OF BORE HOLES PASSING THROUGH WORKABLE BEDS OF COAL.

OFFICERS.

(1) METHOD OF APPOINTMENT.

A chief inspector of gas and oil wells shall be appointed by the governor from an eligible list of men who have passed a satisfactory examination showing their technical fitness for the position.

(2) SALARY.

The yearly salary of the chief inspector of gas and oil wells shall be ——— dollars. Necessary travel expenses to the amount of ——— dollars shall be borne by the State, and an office equipped with suitable filing arrangements shall be provided.

(3) ASSISTANTS.

Assistant inspectors of gas and oil wells, subject to the authority of the chief inspector, may be appointed by the governor.

(4) DEPUTIES.

Competent men may in an emergency be deputized by the chief inspector to perform the duties of his position in the field, but an appeal from the decisions of such deputies may be had to the chief inspector. Deputies may receive not over —— dollars per day and actual expenses not exceeding —— dollars per day.

DUTIES.

(5) LOCATION OF BORE HOLES.

It shall be the duty of the inspector to receive applications for permission to drill bore holes and to issue a license to drill to persons who comply with the law.

The inspector shall receive and file maps giving the location of bore holes, shall determine the sufficiency of such maps for the purpose of accurately locating such bore holes, and shall cause a new survey of such location in case the available maps are unsatisfactory.

It shall be the duty of the inspector or his assistant to examine the maps accompanying applications for permission to drill a bore hole, and if the proposed location is in the vicinity of a mine he shall immediately proceed to the site and request the district mine inspector and the coal operator to state whether the proposed location as indicated on the map accompanying the application to drill is such as will interfere with the safe and economical operation of the mine.

The inspector shall issue a license when the proposed location of a bore hole shall have been determined to be such as will not interfere with the safe and economical operation of any mine or mines that might be affected, as determined by the mine owner, the State mine inspector, and himself. To this end he shall have the power to move the proposed location of a bore hole, as hereafter specified.

After hearing and duly weighing the evidence he shall permit the bore hole to be drilled at such point as will, in his judgment, permit the safe and economical operation of the mine or mines affected.

(6) RECORDS.

It shall be the duty of the inspector to keep a complete record and to prepare for publication a yearly report of the wells drilled in his district, including their location, date of completion, depth, production, and date of abandonment, and the names of owners.

(7) SUPERVISION.

It shall be the duty of the inspector to examine each well in his district at frequent intervals, giving special attention to all wells containing gas in such quantity as, in case of leakage, to make them a menace to a mine; the inspector shall see that all the provisions of this act are observed and strictly carried out.

(8) COMPLAINTS.

The inspector shall receive and investigate all complaints as to injury, present or impending, due to a lack of precaution on the part of any well owner or mine operator, and if he finds the complaints against the mine owner to be well founded, he shall lay the facts before the State mine inspector.

(9) PLUGGING WELLS.

Upon receiving notice of an owner's intention to abandon a well the inspector or his assistant shall proceed to such well and satisfy himself that the provisions of this act referring to plugging wells are complied with. He shall join the owner in making affidavit to the manner in which the well has been plugged.

(10) VIOLATIONS.

If the inspector discovers any well being drilled, operated, or plugged contrary to the requirements of this act he shall order the workmen engaged upon such well to cease work at once and shall not permit the work to be resumed until he is satisfied that the law is complied with.

(11) ENFORCEMENT OF LAWS.

To enable the inspector to perform the duties imposed upon him by this act, he shall have the right at all times to approach and examine any well in his district, and, with the authority of the State mine inspector, to enter any mine affected, and upon the discovery of any violation of this article or upon being informed of such violation, he shall institute proceedings against the person or persons at fault, under the provisions of the law provided for such cases.

In case of failure of the owner to properly plug an abandoned well, it shall be the duty of the inspector to have the work properly performed by contract and to assess the cost against the well owner.

(12) PENALTIES.

There shall be adequate penalties provided to aid in obtaining the safer conditions here proposed.

LOCATION OF BORE HOLES.

(13) APPLICATION.

Any person (firm or corporation) purposing to drill a bore hole through a workable seam of coal shall make application in writing to the chief inspector of gas and oil wells for permission to drill such hole, and he shall not commence drilling until such permission in writing shall have been received by him. In case of dispute the State geologist shall determine whether a seam of coal is workable within the intent of this act.

(14) SURVEY AND MAPS.

Accompanying the application for permission to drill such bore hole there shall be submitted a map, showing the location of the proposed bore hole, with reference by course and distance to the boundaries.

Said map must be made on a scale of 200 feet to 1 inch, and shall be based on surveys made by surveyors or engineers of recognized standing, and shall be certified to by the surveyor or engineer making the same.

In case the inspector finds the map insufficient to enable him accurately and completely to locate the proposed well, he shall require that another survey and map be made before permission to drill shall be granted.

If the original map is subsequently found to be adequate the cost of the second survey and map shall be borne by the State.

(15) BOND.

Any person (firm or corporation) purposing to drill a bore hole through any workable seam of coal shall be required to give bond in the amount of ——— dollars, to insure that the drilling and abandoning of such bore hole shall be in accordance with the provisions of this act.

(16) DISTANCE FROM BUIDINGS, ETC.

No bore hole penetrating a gas-bearing or oil-bearing formation shall be located within 300 feet of a shaft or entrance to a coal mine not definitely abandoned or sealed; nor shall such bore hole be located within 100 feet of any mine shaft house, boiler house or engine house, or mine fan. The proposed location of any bore hole must insure that when drilled it will be at least 15 feet from any mine haulage way or airway.

(17) LICENSE.

If a proposed bore hole be so located as not to interfere with the safe and economical operation of any mine, and if the previous requirements of this act be complied with, a license shall be granted, and said person (firm or corporation) may proceed to drill such bore hole in accordance with the further provisions of this act.

PROTECTION OF COAL BEDS.

(18) CASING OFF SURFACE WATER.

Any bore hole penetrating any workable seam of coal shall be cased by the owner of the bore hole with a suitable casing (conductor or drive pipe), so as to shut off all surface water from entering the coal seam.

(19) CASING THROUGH ANY COAL SEAM.

Any bore hole drilled for gas, oil, or other mineral shall be drilled to a point at least 20 feet below any workable seam of coal that may be penetrated, and receive a metal casing not less than one-fourth of an inch in thickness and of an inside diameter 4 inches less than the diameter of the bore hole. This casing shall be concentrically seated on the bottom of the hole, shall extend to the surface, and shall be known as the first casing.

A second inner casing, 4 inches less in diameter, shall extend at least 10 feet below the first casing and shall be seated in 9 feet of cement mortar composed of 1 part Portland cement and 2 parts sand. The inside of the second casing shall be open to the atmosphere its full length.

The intervening spaces between the second and the first casings and between the first casing and the bore hole or outer wall shall be filled with puddled clay to a height of 30 feet above the coal seam.

(20) CASING THROUGH ACCESSIBLE MINE EXCAVATION.

Any casing that passes through a mine excavation shall be protected by a wall of concrete or of masonry or brick laid in cement mortar, extending from 2 feet below the mine floor to the mine roof. Between the first casing and the wall thus constructed there shall be left an annular space of not less than 2 inches, which must be filled with puddled clay. This work shall be done by the well owner.

Casings that are exposed by mining operations shall be covered by the mine operator in the manner prescribed above.

(21) CASING THROUGH INACCESSIBLE MINE EXCAVATION.

Where a bore hole passes through an inaccessible mine excavation the outer casing shall be protected by cement mortar held within a metal tube with a diameter 4 inches greater than the diameter of

the casing and extending from 4 feet below the mine floor to 4 feet above the mine roof or cave.

(22) CASING TO EXCLUDE WATER.

Before a bore hole is drilled into a gas-burning or oil-bearing formation a string of casing shall be so set as to exclude all water from the lower bore hole.

(23) TUBING A GAS WELL.

To conduct gas from a gas well, tubing shall be inserted with a suitable packer placed below the inner casing and so constructed as to prevent the escape of gas except through the tubing.

The inner casing shall be left open to the atmosphere throughout its full length.

(24) TUBING AN OIL WELL.

To conduct oil from an oil well, tubing shall be inserted and extend from the oil-bearing formation to the top of the well.

The inner casing shall be left open to the atmosphere throughout its full length.

Should gas be liberated in the well in sufficient volume to have commercial value it may be shut in by means of a packer placed below the inner casing and may be conducted from the well through tubing inserted into the well parallel to the oil tubing. (See fig. 11.)

(25) CASING HEADS.

Casing heads must have at least one opening to the atmosphere to which a valve or plug is not attached, so as to insure ample vent in case of leakage into casing spaces.

(26) COMPLETION OF WELL.

When a bore hole has been drilled and put into operation the owner shall file with the inspector a statement of the total depth of the hole, the sizes and lengths of casing used and remaining in the hole, the depth and thickness of all coal seams penetrated, and whether oil, gas, or water is obtained.

(27) SUSPENDED OPERATION.

When for any cause a bore hole that passes through a workable seam of coal shall cease temporarily to be operated, the inner string of casing shall be maintained open to the atmosphere. Should oil tend to rise in the well above the bottom of the first or outermost casing passing through the workable coal seam, such oil shall be

pumped out by the owner, and its level maintained below the bottom of such outermost casing.

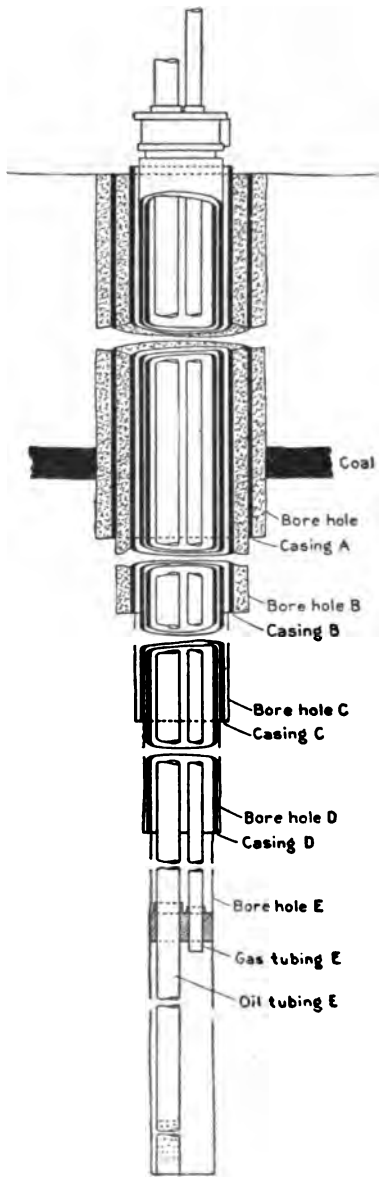

FIGURE 11.—Section of oil well showing tubing for gas.

ABANDONMENT OF WELL.

(28) NOTICE OF INTENTION.

When any oil or gas well is to be abandoned the owner shall notify the chief inspector in writing of such intent to abandon, and shall

proceed with plugging methods only after complete arrangements for inspection shall have been made and permission granted.

(29) METHOD OF PLUGGING.[a]

When a well is to be abandoned it must be entirely filled from the bottom to the surface. The lower hole must first be filled with sand, clay, or rock sediment to a point 5 feet above the lowest gas bearing or oil bearing formation. Above each formation supplying gas, oil, or water, and immediately below any workable seam of coal, there shall be a plug as hereafter described, and each well shall have not less than three such plugs whatever the formation.

The plug above mentioned shall consist of 25 feet of cement mortar made of one part Portland cement and two parts sand properly mixed, and placed with a bottom-dump bailer upon 5 feet of clay resting on a seasoned wood plug 2 feet long and of the same diameter as the hole. Five feet of clay shall also be placed on top of the cement section. The space between the plugs here called for must be filled solidly with puddled clay, sand, or rock sediment, or with cement, mortar, or concrete. The filling shall be complete up to the bottom of a casing before such casing can be drawn; the filling shall continue length by length as the casing is withdrawn, cement, mortar, or concrete being used in those sections of the well hole that pass through coal seams.

(30) LOG OF METHOD OF PLUGGING.

When a bore hole has been plugged, an affidavit in reference thereto shall be made by the well owner, who shall state the method followed, the materials used, and the length of the sections occupied by each material. The affidavit shall be certified by and filed with the gas and oil well inspector. A duplicate of the affidavit shall be filed with the county recorder by the gas and oil well inspector.

(31) RELEASE OF BOND.

After and not until these several requirements have been met, any bond that may have been given by the owner shall be returned.

(32) FEES.

(The question of whether fees shall be charged for license and inspection is left for further consideration.)

DISCUSSION.

Chairman FOHL. Gentlemen, the subject is now before you in its entirety. We have had a general statement of the problem, coupled

[a] See Fig. 7.

with suggestions from Mr. Rice, an account of some of the troubles that have been encountered in mining coal in the vicinity of oil and gas wells by Mr. Jones, and finally the suggestions for laws and regulations by Mr. Hood and Mr. Heggem. In a way the papers form a comprehensive treatise on this subject by the Bureau of Mines, but I think they have made it very clear to us that they claim no finality for the views expressed here nor for the suggestions as to legislation. The subject is now before you for discussion, and I hope that you will all assist in every possible way, whether with criticism or with suggestion bearing on the various points brought out. The meeting is now open to you.

Mr. DE WOLF. I would suggest that we turn these pages slowly, perhaps numbering the paragraphs as we go along, and call for criticisms of the particular pages as we turn them.

Chairman FOHL. I think possibly we might advance matters a little by asking Mr. Hood to read slowly the suggestions, and then stop for suggestions, criticisms, or questions.

Mr. Hood thereupon began rereading the suggestions for laws and regulations in the matter of bore holes passing through workable seams of coal included in his paper.

OFFICERS.

1. METHOD OF APPOINTMENT.

A chief inspector of gas and oil wells shall be appointed by the governor from an eligible list of men who have passed a satisfactory examination showing their technical fitness for the position.

2. SALARY.

The yearly salary of the chief inspector of gas and oil wells shall be ——— dollars. Necessary travel expenses to the amount of ——— shall be borne by the State, and an office equipped with suitable filing arrangements shall be provided.

Chairman FOHL. The first thing we have for consideration is the method of appointing gas and oil inspectors. Are there any remarks on this?

Mr. HICE. It seems to me this is a matter that would vary in the different States very materially. The disposition, in some States at least, is to reduce the number of departments. This suggestion here would practically be making a new department similar to our present mine-inspection department, and in some of the States possibly there would be no necessity for that; in other States there might be. In a State like this, where we have, for instance in 1911, a record of over 3,000 wells being drilled and over 2,000 being abandoned—and how many were drilled and abandoned of which we have no

record is mere guesswork—it would require the appointment of a number of men, and would perhaps justify a separate department, as is suggested here; but probably in most of the States this would go under some of the present organizations.

Chairman FOHL. I think we might ask Mr. Hood to give his ideas on this point.

Mr. HOOD. Under the wording I believe it would be entirely possible for anyone who had satisfactory qualifications, such as the geologists or the State mine inspectors, to be appointed by the governor in whatever way thought desirable.

Mr. DE WOLF. It strikes me that this is a perfectly logical scheme for getting at it. As Mr. Hice has said, the conditions will vary markedly in all the States. In some States, like our own, there will be a civil-service board, which will probably have the responsibility for examining and preparing an eligible list. This is a broad-gage group of men at present in my State (Illinois), and it would doubtless ask for expert advice in holding examinations and seeing that the men were qualified. With reference to the need for a separate group of men, or a separate inspector in my own State, I think that would be desirable, because the coal-mine inspectors there are concerned only with the counties that produce coal at present. We have a number of counties that do not produce coal yet, but will in the near future, and in which at present oil and gas development is at its maximum. So I think it ought to be some one's function to attend to the matter in such counties. He might well be a subordinate under the present State mining board, he might be a subordinate under the State geological survey, or he might be an independent official; however, he should have an acquaintance with both mining and oil and gas well operations, and with the geologic aspect of his work. In other words, I think this is a good scheme on which any State might build up a satisfactory law.

Mr. Hood thereupon read the sections relating to assistants and deputies:

3. ASSISTANTS.

Assistant inspectors of gas and oil wells, subject to the authority of the chief inspector, may be appointed by the governor.

4. DEPUTIES.

Competent men may in an emergency be deputized by the chief inspector to perform the duties of his position in the field, but an appeal from the decisions of such deputies may be had to the chief inspector. Deputies may receive not over ——— dollars per day and actual expenses not exceeding ——— dollars per day.

Mr. CUMMINGS. I would like to inquire who will examine these gentlemen and decide whether they have passed a satisfactory examination.

Mr. HOOD. Each and every State will have its own way for that. As Mr. De Wolf has said, Illinois has machinery for that at present. Pennsylvania, I think, has some sort of machinery for mine inspectors, and might have similar machinery for this. It is not intended to make any detailed suggestion along that line.

Chairman FOHL. It may be entirely possible that some suggestion of that sort would not be out of place. With that in mind, I think I shall ask Mr. Schluederberg to tell us what the machinery of Pennsylvania is for the examination of inspectors.

Mr. SCHLUEDERBERG. I do not know but that the department of mines would be well enough organized and thoroughly competent to take care of gas and oil inspection in Pennsylvania. There is no special attention given to it now by the inspection department. All inspectors of the different districts, of course, to some extent look after wells sunk through mines and see to the safety of miners, but they do not now have any authority to supervise the drilling or abandonment of wells. But as far as I can see, I think that such matters could be handled by the department of mines of the State of Pennsylvania.

Mr. TAYLOR. It appears to me that in my own State (Pennsylvania) it would be better if it were under the geological survey department.

Chairman FOHL. Do you mean the Federal bureau?

Mr. TAYLOR. No; the Pennsylvania geological survey. It appears to me that a part of the survey's duties ought to be the exact location of these wells, with reference to property corners or to monuments, and such work requires a different class of men, and I think probably a different class of examiners to examine the men; and it would be better to have this done under the State geological survey. I am not sure whether the survey now has any way of examining applicants for positions or not, but they could easily be provided for.

Chairman FOHL. We have representatives from other States here, and we would like to hear from them.

Mr. CROCKER. At the present time in West Virginia we are using the United States Geological Survey monuments as a basis to work from. We must have something permanent, and we are using them and making all our locations from them.

In regard to the inspectors, in that State at the end of the year there were 700 wells drilling and probably at the same time 100 or 200 more in operation and abandonment, so the matter of inspectors and deputies is going to be a pretty large problem, because—unless

several wells happen to be pretty close together—they can not visit any large number of wells and do their work properly.

Mr. TAYLOR. I might state that in Pennsylvania we do the same thing. We locate wells on our property from the Government survey monuments, and all the locations are coordinated with reference to the values that we have fixed for the Government monuments.

Mr. HALL. I think the objection to the United States Government monuments is that they are too far away. The surveying done by the oil companies is not, I think, as accurate as that done by the coal companies. In the coal business at one time we had some very poor surveying. To-day our surveys are reasonably accurate. The coal companies are putting in their own monuments for their corners, and by using such monuments as these which are only a short distance from the points of surveying, or by using some other monuments, possibly of a specific kind, we would work out the inaccuracies of extensive surveys. To take the United States Geological Survey points would be a mistake, because it would involve extended surveys that might be extremely inaccurate. The average local surveyor, such as works at oil and gas well locations, is a man who has marvelous ideas both of surveying and of geology. Altogether he is possibly as little competent as any man we could imagine to do that character of work, and the shorter the distance away from the point he is surveying that he has to go for his monuments the more likely we are to get accurate results.

Mr. LAYTON. Then, if there is a company operating in any county, the State geologist could immediately go to that district, and we might be able to get some monuments that would be permanent. As to the question of accuracy, I would take exception to the gentlemen, for the simple reason that in the last few years the location of an oil or gas well is accurately made from actual surveys. There was a time when no attention was paid to that, but all the larger companies to-day have accurate locations of all wells.

Mr. CORRIN. I believe Mr. Moore, of the Consolidation Coal Co. (West Virginia), could tell us how its work is done and the accuracy of the locations.

Chairman FOHL. Would it not be better to confine ourselves to the question first raised here, as we come to the matter of locations later? We are now considering the methods of appointment and the character of the people who are to be the inspectors. Are there any further suggestions in that connection?

Mr. HICE. The question that is brought up here reaches possibly a little further than that. It is altogether probable that any act relating to plugging oil and gas wells under inspection will have to be broad enough to cover all wells drilled, regardless of whether they

are in coal-bearing sections of a State or not. Therefore the matter of the qualification of the inspector is important; he will have to have considerable knowledge of the oil and gas fields, of the different sands, and, in addition, the relations which they have to the coal seams and to the structure. Notwithstanding the title simply refers to oil and gas sands and coal-bearing portions of the State, it is very questionable if an act of that kind will be passed. In getting in our report of production, I have been surprised in the last month or six weeks to receive as many complaints as I have from oil producers, who say that their neighbors will not plug their wells. It is not the first time I have noticed it, but during this week we have had a number of complaints of that kind.

Mr. WATTERS. I think the matter of the location and the parties who do the locating and the matter of inspection might well be taken together, for the reason that as a matter of economy to the State it might be possible that the same individual can perform the same work, but it occurred to me while we were discussing this that we may be conflicting with the already existing 1911 bituminous mining laws, inasmuch as there is a provision in them for the locating of oil and gas wells by the producing companies. It is information I am after, that is all.

Mr. HALL. The main object of these remarks by Mr. Hood, as he has definitely stated, is that there should be some arrangement by which inspectors may be examined. He has omitted to say anything about how the assistants should be examined, and how the deputies should be examined. Of course that could be provided out of the machinery which is already provided by the State, in a general way, for examination, but in making that distinction it more or less seems to help the overlooking of that part, and if that paragraph could be reworded so that the statement might be made that the State government should choose, or at least that those departments should be chosen, from an eligible list of men who have passed satisfactory examinations and shown technical fitness not only for the position of chief inspector but of all inspectors, it might meet my objections. I think that this is simply an oversight in making up the paper.

Chairman FOHL. I notice it is well understood that the various State legislatures will have the last word on this, and I think we scarcely have arrived at the point yet at rewording this paragraph. If that seems necessary a little later, it would be a matter for a committee, but I should think we should go just as far as possible in the matter of suggestions, to give such a committee such suggestions as Mr. Hall has just made, and if there are any further suggestions along this line we will be glad to hear from them.

Mr. RICE. That is really the object of presenting merely a skeleton set of rules and regulations. We found that there was such a com-

plexity of matters that we could not hope to make more than a general outline. It was thought that a committee which might be organized by this body could gather together such suggestions as have been made and might then be able to put them into shape. One could go into almost endless detail in the attempt to provide just how the several service bureaus, for example, should examine deputies and others. The outline was merely intended to be suggestive. It was not intended in any sense to be a final draft. We thought the committee would be much better qualified to make such a draft.

Chairman FOHL. At the same time, there is no intention to make you feel that you are hurried. Any further suggestion will be welcome. If it gets to the committee stage such work as this will be very helpful to that committee. If there are no further suggestions in connection with the personnel of the inspectors we may proceed to the matter of the duties of the inspectors.

DUTIES OF OFFICERS.

Mr. Hood thereupon read the next section, as follows:

5. LOCATION OF BORE HOLES.

It shall be the duty of the inspector to receive applications for permission to drill bore holes and to issue a license to drill to persons who comply with the law.

The inspector shall receive and file maps giving the location of bore holes; shall determine the sufficiency of such maps for the purpose of accurately locating such bore holes; and shall cause a new survey of such location in case the available maps are unsatisfactory.

It shall be the duty of the inspector or his assistant to examine the maps accompanying applications for permission to drill bore holes, and if the proposed location is in the vicinity of a mine he shall immediately proceed to the site and request the district mine inspector and the coal operator to state whether the location on the map accompanying the application to drill is such as will interfere with the safe and economical operation of the mine.

The inspector shall issue a license when the proposed location of a bore hole shall be such as not to interfere with the safe and economical operation of any mine or mines that might be determined by the mine owner, the State mine inspector, and himself.[a] To this end he shall have the power to move the location of a proposed hole, as hereafter specified.

After hearing and duly weighing the evidence he shall permit the bore hole to be drilled at such point as will in his judgment permit the safe and economical operation of the mine or mines affected.

Mr. DE WOLF. It is doubtless true that some oil and gas operators at certain times want to proceed with all possible speed in starting their operation. I am wondering whether the requirement that the chief inspector shall proceed to the spot and notify the district mine

[a] The authors of this paragraph agreed that this should have read "as determined by the gas and oil well inspector after due hearing of evidence offered by the mine operator and the mine inspector, as well as by the drilling applicant. (See discussion following.)

inspector, the coal inspector, etc., does not need attention in that regard. Now, in most States, if conditions are anything like those in Illinois, it will take some time to get the district mine inspector on the ground. He has other duties, and unless these duties are considerably reduced from what they now are, it will be a matter of some days before he can be present at the spot where it is proposed to drill the well. I am wondering whether the gentleman would think it altogether necessary for that district mine inspector to be present, or whether the oil inspector, together with the representative of the coal operator and of the oil company at the place, could pass on this question as to whether the location was a safe one.

Mr. RICE. We thought that the parties interested should be notified. If the case is an important one, then they would be present, but if they are not present, then the gas and oil well inspector could at once proceed and either approve the location of the hole or disapprove of it, as the case might be. The thought was that there are three parties in interest—the operator of the well, the coal-mine operator, and the miner as represented, with respect to safety, by the mine inspector. If it is a case which appears at all dangerous, it will certainly be the duty of the inspector to be present. If he is not present, it will be understood that it is a case in which he feels there is no particular danger. The same will be true as far as the coal operator is concerned, perhaps. If he does not choose to be present, it will be presumed that the location of the hole of which he has received notice is satisfactory to him.

Mr. LELAND. That is the idea, and that should be stated very clearly in the statement of the duties. The next paragraph states that the State mine inspector and the State oil inspector must agree. It makes it mandatory in this case. Is it the idea that if one does not come the others can go ahead? If so, we should say so plainly.

Mr. RICE. It is true that there is an apparent conflict; this came about because after assembling our ideas we did not have time to harmonize them fully. It was the intent to give the gas and oil well operator the power to decide after he had given proper hearing to the several interests.

Mr. CROCKER. It seems to me that there is a party left out—the landowner. He gets a royalty, if it is oil or gas. I think he should be recognized.

Mr. TAYLOR. I think the oil man is hardly interested until he has permission from the surface owner.

Mr. CROCKER. I think he is decidedly interested.

Mr. MOLYNEAUX. I think the intention of the entire discussion is particularly to protect the miner and the mine owner largely. The oil man and the gas man, of course, will have to bear largely the brunt of what is going on, as far as protection goes, and I think that

in some cases it has been suggested that, for instance, a gas man would get a lease which limits him to a certain time in which to drill a well. Now, he may get a good well if he can do the work within that time; but if he has to wait until all these people are seen and get their permission, the lease may expire and he will lose out. Those are instances within my knowledge. And while fair to the mine owner, the regulations should also be fair to the driller. After the driller has duly notified the mine owner and talked to the inspector of that district, if within a certain fixed time, a few days, they do not pay attention to it, he should be permitted to go ahead with his work. I think there is no provision of that kind intended in the article, but there should be something of the kind in order that the man who drills the well, or has his money invested in that business, and is looking forward to possible success with a certain well and with other wells in the immediate vicinity, may be able to go ahead within a reasonable limited time and drill his well after he has notified this inspector and also the mine owner.

Mr. HOOD. Let me say that on reading this section over the second time I am quite ready to admit that it is not as clear as it should be. It does not bring out the intent as sharply as it should. It is believed that if there were some such law as this then even to a greater degree than is true at the present time these parties would get together beforehand, and it would only be in a rare case that all of this machinery would be brought into play, for the parties would agree in short order, and the inspector would have no formal meeting of this sort. That is the intent, and it can be brought out better and much more clearly with different language.

Mr. LAYTON. I suggest that it be amended to read something like this: File the paper; and if within 5 or 10 days they fail to appear, then let the party drill his oil or gas well and proceed immediately.

Mr. MOORE. A coal operator might not know about that as well as the gas-well operator. The coal operator, unless some other provision is made, ought to have some way to know that this location is made. On the other hand, the gas-well operator would have no way of knowing whether the well would interfere with the mine or the mine workings unless he has access to a mine map. He has got to communicate with the coal operator before he knows whether the location is objectionable or not.

Mr. DE WOLF. This is just a matter of routine, after all. If there is a State gas-well inspector, or somebody acting for him in his absence, a request or application which comes in would then be duly forwarded and a time limit put on it. Forwarded by registered mail, if you wish, notifying the coal operator, the mine inspector, and anybody else who is in interest in the matter; and after the lapse

of a certain length of time let it be understood that the oil and gas man, if there has been no objection, can go ahead.

Mr. TAYLOR. This matter of notice, it appears to me, could be confined to well locations over an active mine, a "live" mine, and where that is the case there ought not to be any boring until the location has been actually passed upon. It is not simply the driller then. It becomes a question of life and death. This clause here, it seems to me, is definite in that.

Mr. CORRIN. The company that I am interested with has a system of locating that has proved quite satisfactory. When we make a location we make an additional map, giving the measurements of that location, that is, where that well is to be drilled. When we send a record of the location out to the field man we send a duplicate to the people who own the coal under that land. Of course, we have an abstract made of the title to the land before we make a location, and we know how that land is platted, and before any injury is done by drilling the coal owner comes back with his suggestions. I do not see why an addition of that kind could not be made to include the coal operator. If the coal operator has any suggestions he can take it up to the inspector.

Mr. HOOD. What would be a reasonable time, in your judgment, for the approval of the location for a well?

Mr. CORRIN. I should say a week. A week would be the outside limit, in my judgment.

Mr. RICE. As Mr. Hood has indicated, in giving this outline we have merely suggested the machinery for handling a case. We expect that in the greater number of times the matters will all have been arranged beforehand between the well operator and the mine operator, before it even goes to the chief oil and gas inspector, if there is such, and it is only in the exceptional case that there will be any difficulty, but when such arises the machinery is provided for handling it.

Mr. HICE. Immediately upon an application, then, the requirement would be for a plot with sufficient references to actually locate the well, and that would be a part of the application. And undoubtedly that should also contain a statement as to whether the well is located over a mine or near a working mine, and the inspector would at once have that information, and if the blanks were properly prepared he would know exactly to whom the notice was to go, and if the location was over a working mine, part of the requirements would be to show the position of the mine workings.

Mr. CORRIN. The larger coal companies furnish maps of their mine workings, and in making our locations we try to keep away from very bad spots.

Chairman FOHL. I take it that the consensus of opinion is that all three parties in interest must have notice, but that a certain time must be set, so that after they have had notice it is possible to proceed. Are there any further suggestions along this same line? The chair would like to ask who is to furnish the maps that the inspector is to receive and file? What is the purpose of that paragraph?

Mr. HOOD. This is only defining the duties of the inspector. Where the duties of the oil well and gas well people are taken up, is made clear.

Chairman FOHL. Suppose you take the next three subjects together.

Mr. Hood thereupon read the next three subjects.

6. RECORDS.

It shall be the duty of the inspector to keep a complete record and to prepare for publication a yearly report of the wells drilled in his district, together with their location, date of completion, depth, production, date of abandonment, and name of owner.

7. SUPERVISION.

It shall be the duty of the inspector to examine each well in his district at frequent intervals, giving special attention to all wells containing gas in such quantity as, in case of leakage, to make them a menace to a mine; the inspector shall see that all the provisions of this act are observed and strictly carried out.

8. COMPLAINTS.

The inspector shall receive and investigate all complaints as to injury, present or impending, due to a lack of precaution on the part of any well owner or mine operator, and if he finds the complaints against the mine owner to be well founded he shall lay the facts before the State mine inspector.

Mr. DE WOLF. In the sixth paragraph you mean each active well, do you not?

Mr. HOOD. Each active well.

Mr. HALL. The name of the owner—not only the owner of the land, but the owner of the coal—the operating company that may own the coal, and the name of the person on whose farm the well is to be drilled, so there may be some way to trace records and look up the incumbrances that may be on any farm where there are gas wells or oil wells.

Chairman FOHL. Are there any further suggestions in reference to this matter?

Mr. HOOD. The general subject of penalties has not been entered into here at all, but when this goes into a lawyer's hands he would doubtless add the penalties to make this workable.

Mr. Hood then read the provision relating to plugging wells.

9. PLUGGING WELLS.

Upon receiving notice of an owner's intention to abandon a well the inspector or his assistant shall proceed to such well and satisfy himself that the provisions of this act referring to plugging wells are complied with. He shall join the owner in making affidavit to the manner in which the well has been plugged.

Chairman FOHL. Are there any remarks on this subject?

Mr. HALL. When an undertaker buries a man he has got to have permission to do so, and it seems to me that when a man plugs a well not only the owner ought to receive permission, but the man who does the plugging ought to be made responsible, and he ought to be required, or at least to be held liable if he has not received that permission before he does the work, just as the undertaker law is in our State—a man who buries is responsible if he has not received permission from the board of health. In this case the man is responsible who does the plugging if he has not received the certificate that the well may be plugged and that notification has been duly given.

Mr. RICE. Paragraph 11 makes provision for this.

Mr. MOORE. I should like to ask one question. Suppose this oil-well owner makes his application to the gas-well inspector to plug a well. Suppose the inspector is present and signs this certificate, but suppose, nevertheless, that something happens from that well—that a miner is killed in the mine. The miner's family will bring suit against somebody. Who is to be held responsible? Does the State assume the responsibility on account of its representative approving the method, or are the gas people or the other people to be held responsible, or both?

Mr. LAYTON. If the accident was caused by delay on the part of the coal operator or the mine inspector, I suppose they would be liable.

Mr. MOORE. The part I wish to bring out is that the present responsible oil and gas companies feel that they have a moral responsibility for taking care of the wells that they plug, and because of that moral responsibility they plug them as best they know how. Would this be a loophole for somebody that doesn't want to spend quite so much money in plugging wells to release him from the present laws?

Mr. HOOD. It doesn't seem to me that such release would follow from a law of this sort. This is simply a supervision—a police duty—to see that they do the best they can, and yet it would not release one from responsibility. I think we have similar cases with other laws.

Mr. CAMERON. Going back to the paragraph before that, under "Complaints," is the State mine inspector the referee or the final

arbitrator in this case? It says he shall lay the facts before the State mine inspector.

Mr. RICE. That was intended to meet a complaint against the coal operator. The intent of it is this: That the gas and oil well inspector, if there be such, would have supervision, as far as the gas and oil well operation is concerned, but if through mishandling within a mine damage is done or anticipated and complaint has been brought to the attention of the oil and gas well inspector, he would not have the power to enforce within the mine, that would be the State mine inspector's duty, and oil and gas inspectors should simply lay the facts of the complaint before the State mine inspector. He will then have fulfilled his duty, and it will be up to the mine inspector to investigate and remedy——

Mr. CAMERON. Then, as the State mine inspector is subject to the department of mines, you will have a conflict of law.

Mr. LAYTON. Where would the oil or gas well operator be? You make no provisions for him.

Mr. CAMERON. He would be subject at all times to the law that might be adopted. There is no law at this time governing oil and gas, but the coal operator is already provided for by the mining laws.

Chairman FOHL. Are there any further remarks? If not, we will take up the matter of the enforcement of the law.

Mr. Hood read the provision relative to violation and the enforcement of the law.

10. VIOLATIONS.

If the inspector discovers any well being drilled, operated, or plugged contrary to the requirements of this act, he shall order the workmen engaged upon such well to cease work at once and shall not permit the work to be resumed until he is satisfied that the law is complied with.

11. ENFORCEMENT OF LAWS.

To enable the inspector to perform the duties imposed upon him by this act, he shall have the right at all times to approach and examine any well in his district, and with the authority of the State mine inspector, to enter any mine affected, and upon discovery of any violation of this article or upon being informed of such violation, he shall institute proceedings against the person or persons at fault, under the provisions of the law provided for such cases.

In case of failure of the owner to properly plug an abandoned well, it shall be the duty of the inspector to have the work properly performed by contract and to assess the cost against the well owner.

12. PENALTIES.

There shall be adequate penalties provided to aid in obtaining the safer conditions here proposed.

Mr. WATTERS. A good many people are considered in formulating laws of this kind, but how about the man who goes to his neighbor

or by himself employs a neighbor to drill a well? He has enough means to finish that well, but, because of the plugging on account of not getting gas or oil, he does not have enough money. It seems to me there ought to be something put in as to a bond by individuals, for there is more trouble with them than with the corporations. We have no trouble with the largest corporation; it is with the individual who drills a well to make something out of it or "bust". There ought to be something to cover that individual.

Mr. HOOD. That is covered a little later by a paragraph that requires a bond to be put up. If a man hasn't got enough money to drill a well and to pay for the bond, he doesn't drill.

Mr. PAUL. In connection with the section just read I should like to ask if it would apply to all wells being drilled in new territory, where the operators of the well are desirous of keeping secret whether or not the territory is productive; or is there anything in here against giving out advance information in connection with the authority given the inspector to examine any well at any time?

Chairman FOHL. I suppose we might say that it is to be taken as a matter of course that information of that kind is confidential and must stay with the operator and not be spread abroad, but I presume when we come to make a final act it ought to be mentioned there.

Mr. HALL. It seems to me the depths at which oil is to be found are so great that as far as drilling is concerned his (the inspector's) interest might cease after the well is through the coal measure. But when it comes to plugging, that is a different proposition.

Chairman FOHL. I think the coal operators are interested in the well until it is finally abandoned.

Mr. RICE. We should like to hear from gas and oil well people on this. On the matter of plugging I think that the provisions of a legislative act should be comprehensive. The oil and gas well operators, I understand, would like to insure that the plugging of every hole is properly done to prevent salt water leaking from one horizon to another. In order that this be efficiently done, it would seem necessary that the inspector have full records of the hole from top to bottom.

Mr. MOLYNEAUX. The actual value of a well should not be disclosed by the permission to drill, or by any inspector. The records asked for should not say anything about the value of the well. It is simply a matter of safeguarding, and I do not think the matter of value should be disclosed. Assuming that the inspector would know enough to be discreet in such matters, I do not think the matters of secrecy should or would be disclosed by inspection, unless the inspector was a grafter, and we assume that men in such positions should be above graft.

Chairman FOHL. Mr. Corrin, would you care to say anything in reply to what Mr. Rice has just suggested?

Mr. CORRIN. No, sir; I do not think there is a great deal of danger of disclosures. There might be a case once in a long while where there might be trouble. I do not believe all the information you would learn would do anybody any good or harm.

Chairman FOHL (to Mr. Rice). Don't you have in mind the matter of plugging between various sands?

Mr. RICE. That is what we intend to be a matter of public record, so that it might afterwards be accessible, and if the oil and gas inspector did not have access at the time of drilling it would be a difficult matter to get the information later. As far as giving out information is concerned, it is entirely conceivable that an inspector might not do his duty, but as far as I know, and I have considerable acquaintance with coal-mining inspection, it has rarely happened that information has been disclosed which was injurious to the operating company. I think the inspectors are generally reliable, and I do not think we ought to anticipate the contrary. It is manifest that if any inspector failed to do his duty he would promptly render himself liable to removal.

LOCATION OF BORE HOLES.

Chairman FOHL. Suppose we take up the entire next five sections, 13 to 17, inclusive, from the matter of locating to the protection of coal measures, as they all seem to be interwoven.

Mr. Hood thereupon read the section suggested by the chairman.

13. APPLICATION.

Any person (firm or corporation) purposing to drill a bore hole through a workable seam of coal shall make application in writing to the chief inspector of gas and oil wells for permission so to do, and he shall not commence drilling until such permission in writing shall have been received by him. In case of dispute the State geologist shall determine whether a seam of coal is workable within the intent of this act.

14. SURVEY AND MAPS.

Accompanying the application for permission to drill such bore hole there shall be submitted a map showing the location of the proposed bore hole with reference by course and distance to the boundaries.

Said map must be made on a scale of 200 feet to 1 inch, and shall be based on surveys made by surveyors or engineers of recognized standing, and shall be certified to by the surveyor or engineer making the same.

In case the inspector finds the map insufficient to enable him accurately and completely to locate the proposed well he shall require that another survey and map be made before permission to drill shall be granted.

If the original map is subsequently found to be adequate the cost of the second survey and map shall be borne by the State.

15. BOND.

Any person (firm or corporation) purposing to drill a bore hole through any workable seam of coal shall be required to give bond in the amount of ——— dollars to insure that the drilling and abandoning of such bore hole shall be in accordance with the provisions of this act.

16. DISTANCE FROM BUILDINGS, ETC.

No bore hole penetrating a gas-bearing or oil-bearing formation shall be located within 300 feet of a shaft or entrance to a coal mine not definitely abandoned or sealed; nor shall such bore hole be located within 100 feet of any mine shafthouse, boiler house or engine house, or mine fan. No bore hole shall be located to drill within 15 feet of any mine haulage way or airway.

17. LICENSE.

Should the proposed bore hole be so located as not to interfere with the safe and economical operation of any mine, and if the previous requirements of this act be complied with, a license shall be granted and said person (firm or corporation) may proceed to drill such bore hole in accordance with the further provisions of this act.

Mr. HALL. In section 16, I wish that word "abandoned" was followed by the words "and sealed."

I was interested in the Consolidation Coal Co.'s mine, and the interest the company showed in a particular mine that was opened near a gas well. The company was anxious to know whether any of that gas would be drawn into the mine. I think in many cases that care is not taken. There may be a mine opening not far from a gas well. The well may not be properly sealed or it may not be sealed at all, or it may be sealed at the opening, and there may be a break in the surface above and the air drawn may be at any time contaminated by the leakage from that gas well. Two responsible gas operators have made careful surveys and their work, which I have noted myself, agrees very closely with the coal-mine surveys which have been made. But I wish to say these two companies are not completely representative of the field. There are some people, "wild catters," whose surveys are not at all reliable and whose locations are possibly made with a flexed stick. It is people like that against whom we have to be on guard. The arrangements in certain parts of West Virginia are creditable both to the oil producer and to the coal corporation, but there are sections where the surveys are creditable to neither, and it seems to me that one of the requirements which we should make is that if the State is not going to make these surveys the connection shall be made to two monuments and those monuments connected. Then we shall have a survey that can be traversed and the State or anybody else who is interested could ascertain at any time whether it was correct or not. * * *

Of course, the United States monuments would be desirable, but if the distance is very considerable to where that survey has to be extended it would be undesirable. It might work very well with large corporations doing a great deal of drilling, and naturally, therefore, in portions of the fields not far from some monument of the United States Geological Survey. But with a small "wild catter" it might not only put a considerable burden on him, but the survey when made would be inaccurate and of no value.

Mr. RICE. I might explain what our intent was in saying that "no bore holes penetrating gas and oil formation shall be located within 300 feet of a shaft or entrance to a coal mine not definitely abandoned or sealed." In the preliminary conferences we had quite a discussion as to drilling through old, abandoned mines, and some of the gas-well people said it was very difficult at times to determine whether a mine is definitely abandoned. It might have been an old country bank and might not have been sealed. It might not ever have been definitely abandoned; that is, no notice of abandonment was ever given. We thought, therefore, that we should leave the wording in rather general terms, putting it up to the gas and oil well inspector to determine what steps should be taken.

Mr. HOOD. In regard to the survey, it was intended to tie this up pretty tight, a good survey being recognized as absolutely necessary, and it was thought that if the courses and the distances to the boundary were given you could not locate two boundaries without having three monuments—three points from which to locate the boundaries—and that such a provision would still be flexible enough to cover a larger number of cases than a provision that surveys should be carried to the section corners or the Geological Survey monuments, or something of that sort.

Mr. LAYTON. Assuming that we are about to locate a well in what is known as "wild-cat" territory. On the filing of that application then the mine inspector or the oil and gas inspector could immediately place two or three monuments, which should govern all future developments in that neighborhood. That would eliminate all further trouble.

Mr. CAMERON. In section 16 it states, "No bore hole shall be located to drill within 15 feet of a mine haulage way or airway." What is the intent of the provision? Does that mean solid coal or in a cut-through or in an abandoned working or gob, or what?

Mr. RICE. The intent is to cover the situation where the gas and oil people have rights prior to the coal operators, and therefore demand that they be enabled to locate a hole anywhere. We have to recognize that in most cases the two interests will get together and

make mutually satisfactory arrangements; but there may be cases where you have to provide legal machinery, matters in regard to which there is a disagreement; that is, it may be that the coal operator will say it is dangerous to locate anywhere in or near a mine or a gas well, which would prevent the drilling of a well on the property. I think we have got to recognize the rights on both sides. The thought was that where the gas and oil well company had those prior rights they should then give the oil and gas inspector authority under the police power of the State, on account of safety, to prevent the hole being drilled through a haulage way or airway, which would include either an intake or a return airway. It would require that it be drilled to one side. The distance 15 feet was chosen merely because we thought that would take care of the errors of a survey or in drilling, in locating a hole enough to one side to permit mechanical protection.

Mr. CAMERON. That had reference entirely to the errors in the survey, and not to the location of the hole?

Mr. RICE. To provide that there should be no hole actually drilled through a haulage road or an airway, or so close that it could not be protected by a covering without interfering with the haulage road or airway, we agreed, as a sort of compromise, on not allowing the hole to be started at the surface nearer than 15 feet from one side or the other, as indicated by the survey. There might be errors in the survey or the hole might not go down truly vertical. We considered that between errors of survey or lack of verticality the hole might be off the true position in striking the coal bed 5 or even 10 feet. It would allow you to put a hole down in the middle of a chain pillar or on the other side of the entry in a barrier pillar or even in a room.

Mr. CAMERON. Your gob or fallen strata might be only 15 feet, or it might be a good many more feet away from that.

Mr. RICE. Yes; we did not attempt to frame a clause preventing drilling through broken ground or abandoned areas, because the courts, at least, have already decided that gas and oil operators who have the property rights have the right to drill through such areas where protection that in the eye of the court is adequate is given.

Mr. HALL. I would suggest the addition of the words "made or provided" at the end of the statement "haulage way or airway."

Mr. RICE. This matter of 15 feet, or any other distance that a location may be moved by an inspector, should be very carefully considered by the coal operators present. We realize that it is a vital point and a very important one in any determination of the powers given to gas and oil well inspectors. It would be applied only where there has been a nonagreement between the two sides.

Chairman FOHL. I think we might come back to this matter of distance after a thorough dicussion of the protection of coal measures, which is the next general topic.

Mr. DE WOLF. May I raise a question here for the guidance of any committee which attempts to work out the details? This section with reference to the location, where it speaks of any person proposing to drill a bore hole through a workable seam of coal, etc., "In case of dispute the State geologist shall determine whether a seam of coal is workable within the intent of this act." I want to point out that you are putting a very difficult problem up to the State geologist. What is a workable coal seam? Is it a coal seam workable 5 years hence, or a coal seam workable to-day? We have in Illinois a law requiring that when wells are to be drilled through a workable coal seam, certain records shall be filed with the State mining board and a certain procedure shall be followed. It does not say who shall decide whether a certain seam is workable; if it did, it would be very difficult for any one to say how thick a seam actually is from the record obtained with a churn drill. If one knows or thinks he knows, it is a question whether his judgment is to be trusted as to whether it is workable now or will be workable in the future. There have been drilled in Illinois over 20,000 wells in a region where there is no active mining, and most all of the gas well drillers will say to you, "Oh, no; we didn't go through any workable coal; there is no coal at all," and yet the diamond drill going down in that precise territory has shown that there are two workable coal beds which some day will doubtless be worth mining. So it seems to me that it would be essential in our State that this permission should be obtained and these regulations observed, for all wells drilled, certainly all within areas underlain with coal. Moreover, proper plugging of a well is certainly important to the oil and gas people in protecting their sands from the water of a wildcatter who is irresponsible and it would be to their advantage to have this section include all borings for oil and gas.

Chairman FOHL. Within the coal measures?

Mr. DE WOLF. I should say that anywhere within the State it would be to their advantage to have these regulations followed.

Mr. RICE. We realized that we are putting a difficult problem to the geologists. We didn't know any persons better able to judge than the geologists. We did consider the use of the words "coal measures," but that is more indefinite for our purpose; for instance, the lower coal measures may not contain workable coal. We thought that by including only the workable coal we could confine it within the boundaries of those coals which could be perhaps in the most remote terms considered as workable; that is, if you have

the lower measures carrying a coal which in no place has been found to be over a foot in thickness, it is a negligible chance that it will be worked, so for practical purposes could be disregarded. Our aim has been to minimize the expense to the gas and oil well operators, where safety or conservation was not at stake; that is to say, where it is clearly without workable coal areas, there need not be delays in getting permission to drill.

Mr. HOOD. I think there is one other thought there, too; that is, it might be entirely proper to lessen the requirements when there is a lower coal bed which in some other districts is workable, yet is so deep that within 10 or 15 years, or within the probable life of a well in that district, it is quite unlikely that the coal in that locality will be mined. It seems very difficult to pick words that will allow drilling without restriction in one part of the county and not in another part, when permission depends on the judgment of some qualified person, and it seemed best to put that responsibility in the hands of the State geologist. The idea was to make it easier for the gas and oil well man, who would not have to case down to a deep-lying coal bed, where it probably would not be mined within the life of that well.

Mr. DE WOLF. Would it not be well to require that the application for a permit shall be filed in all cases, so that the man in charge or the State geologist might inform the parties as to what the likelihood of going through coal strata is? In other words, the first opening sentence says, "any person purposing to drill a bore hole through a workable seam of coal shall make application." He may not know; he may be quite ignorant as to whether he is going to go through a coal seam. He first ought to make inquiry or file an application, and give some one a chance to tell him whether he is going to go through a workable coal seam.

Mr. HALL. You lay too much emphasis on coal. I have had some experience in clay. I know of a 30-foot bed of clay being removed, and a cave in that 30-foot bed came so close to an oil derrick that the sills were hanging over the edge of the hole. We do not want to lay too much emphasis on coal. The idea is to protect all mines; iron mines perhaps need not be considered, but at the same time there are large deposits of clay in Pennsylvania which are close to the gas deposits and the oil deposits.

Mr. RICE. I should like to hear from Mr. Hice as to how near the gas and oil areas are to the underground limestone quarries in this State, with which I am not familiar.

Mr. HICE. As I recall, in the State at least two companies with underground limestone workings are working where oil and gas are produced. There is no question about the fact that the regulation ought to apply in all cases, and there ought to be a broader statement

than one covering simply coal mining. We do not think much about any mining other than coal mining in this part of the State, but other kinds of mines, as any person looking up the production of minerals in the State of Pennsylvania will find, are numerous. There are clay mines in almost every county where coal is produced, and likewise in some of the counties that are not producing coal in commercial quantities. We have very valuable clay products.

Mr. HOOD. I should like to ask whether, if this were made to apply to all bore holes, it would act as a hardship on the oil and gas industry, or would it not? What is the opinion on that?

Mr. HICE. Probably half of the oil wells in Pennsylvania are in coal-producing counties. The other half are in what we class as noncoal-producing counties. I do not know, but personally I think we should require applications to drill in all counties in the State of Pennsylvania. There are other interests to be conserved and looked after in this matter of drilling. The matter of application or location is simply a part of the whole scheme, and the preservation of the oil and gas is to be looked after in the noncoal county as well as in the coal counties; and there is the matter of the pollution of surface water, which is important in every county, and is of very great importance in some counties. I know, for example, that the water in a certain river several times has been so badly polluted from the oil wells in Potter County that the water could not be used for boiler purposes. But water for domestic use is a matter that should be kept in consideration all the time in considering this matter of regulating the location, operation, and abandonment of each oil and gas well and each small water well also.

Mr. CROCKER. As to the pollution of water by oil and gas wells, I think that is pretty far-fetched when we see what has been done by the coal mines. Nearly all the larger oil and gas companies are trying to do all they can to make everything safe, but I think the general tendency is to make us the scapegoat. The whole weight is thrown on us. The landholder has got to be taken into consideration. He has an interest in this matter, and perhaps the man getting a royalty from the coal has a right to say something. I think the whole question of the danger from wells is exaggerated greatly. Of the two cases cited here this morning, the one of the Consolidation Coal Co. concerns something that can be easily overcome. It was simply an oversight that should never have happened. With care it never would have happened. In both cases it was through an oversight, and, unfortunately, those two cases have made the question of danger to appear much worse than it is.

Now, answering for the company I am connected with, we are willing to do everything we can to make it safe, and I think our

relation with the coal companies lets them know that we are carrying out everything we can to do it.

Mr. TAYLOR. I can speak with reference to the mines of the Pittsburgh Coal Co., and of the men who drill for gas and oil through its territory. There are a great many of them doing everything they can, everything within reason. There are others that will not do anything. As to the danger, I will say that at the Dunbar mine, where miners cut into a bore hole, there were 29 men killed. That is not cited in the records. It is really dangerous. Every mining man knows that it is dangerous, and he is afraid of the situation.

Mr. CORRIN. To my mind, it is the future and not what has been done for the coal; the future is more important than something about wells already sealed. Miners are likely to run into an old hole at any time. It would be of more benefit to know where existing wells are located, than the future wells.

Mr. LAYTON. I think the idea is that we should do everything possible to conserve the oil and gas and at the same time to save the men. The object of this meeting is largely that we can do everything possible and reasonable to be done. Speaking about Butler County, the slope mines are all gone. There is no danger of running through them, and I think we all feel that way.

Mr. LELAND. The language of this application refers to bore holes, and it might mean simply bore holes for a shallow water well. If it is to be for all bore holes, it is a hardship on the people drilling small water wells, all over this State.

Chairman FOHL. Are those water bores not a menace to the mines as well as others? Isn't the case Mr. Hall cited one of that sort? It is the column of water standing in the well that is liable to cause an accident, so that these shallow holes might have their own dangers as well as the deep ones.

Mr. LELAND. Is it the thought that any well driller drilling a well must make an application before he is to drill a hole?

Chairman FOHL. If he drills through a coal mine.

Mr. HOOD. If it is to go into a coal seam, or the likelihood is that it would go into a coal seam, he ought to specify not only what the hole is for but where it is and what it is likely to strike, no matter whether it is a water well or an elevator hole, or whatever it may be.

Chairman FOHL. Or whether it is for testing for coal?

Mr. HOOD. Yes. Then it ought to be under supervision in some way. There may be limitations, but it is the intent to cover any hole which might endanger a mine.

Mr. LELAND. That is just what I want to bring out. I put a water well down 125 feet and went through two small seams of coal. If that is the case everywhere, the same thing would apply everywhere.

Mr. HOOD. Were they workable seams?

Mr. LELAND. That is a matter of judgment.

Mr. RICE. I am not altogether in accord with my colleague in the matter of trying to cover either shallow wells or prospecting holes drilled only to the horizon of a workable coal bed, with the intent of prospecting the coal bed, or possibly in certain cases of furnishing a vent for gas from that coal bed. I think that we had better not attempt to regulate the very large class of holes that have a very different purpose. It seems to me we had better confine the suggested regulations to the deep wells that pass below the workable coal beds. That is my personal view.

Mr. HOOD. I know there was some limitation in the discussion. When we were told that producing wells of good quality had been struck within 100 feet of surface, and some water wells would have to go considerably more than 100 feet, and that either of them might go to or through a coal seam which is now being mined, it seemed difficult to pick words that would differentiate one from the other and protect the situation. So it was put in this form. I think that was the last idea.

Mr. RICE. While I appreciate there are some dangers from shallow wells and also that there are some productive gas and oil wells in the upper Carboniferous series, yet I think the chief dangers to mining come from gas and oil in the sands below the workable coals, and when you pass a measurable distance, say 30 or 40 feet or more below the lowest workable coal seam, a bore hole then becomes one which ought to be covered. I feel it is very doubtful whether we should attempt to cover all kinds of drill holes by the proposed regulations; it would be a complicated subject. The matter of record of all holes would be desirable from the public standpoint, but, on the other hand, it might cause serious trouble in the case of prospecting for mineral to make the records public.

Chairman FOHL. It seems a little difficult to make fish of one and fowl of another.

Mr. CAMERON. Regarding the protection of the wells, we have had some experience with wells such as Mr. Taylor mentioned. The drillers did not always do what they agreed to do. When we deal with good people you get everything done you could want, but we have had some dealings with people that wouldn't do even what they agreed to do. In 1906 our mine No. 3 cut into an abandoned gas well containing 400 feet of water. The well didn't do any damage, but it frightened the men who did not know it was there. We then made an agreement with the gas company that any wells drilled in the future would be covered by an agreement. The company drilled a well in 1908, and complied with the contract. It wanted to drill a well in 1910 and selected a location, and we told the company that

location would not do, as it went down in our gob. The company said it didn't care; it would look out for that all right. It went through the gob, and when it attempted to carry out the contract, that is, to cement between the casing and the wall of the well, it could not. All the cement went out in the old gob, and the company applied to the court for a release from the contract. In some inconceivable manner the well was filled up and at once the company went into court and withdrew its bill. Then we went into court and filed a bill and asked that the company comply with its contract, which we considered it had not done. We did not get the release we expected in court. We had the testimony of five mine inspectors that that well was a menace to our mine. It was near an intake airway and connected with a room from a butt entry, and the intake airway will certainly draw the gas from that oil well into our mine. We applied to the State department, and threw the burden on it to stand between us and the gas company.

Mr. LAYTON. My idea is this: The oil and gas people all need protection; everybody needs it; the farmer needs it; he must be taken into consideration as well as the oil and gas operator and the coal operator. All we want to do is to be able to get the oil or gas and at the same time be able to put that farm owner to the least inconvenience possible and still safeguard the life of the man who is mining the coal. That is the problem we have, and the only one, and the one that we must take care of. How we can do that and still put no great burden on the oil and gas man and let the coal man get his product is the question.

Mr. CAMERON. We never denied the right of the gas company to get the gas. We want them to get it, under a certain restriction. I understand that to be the intent of this proposed set of regulations.

Chairman FOHL. We understand that is the purpose. Are there any further remarks?

Mr. SMITH. Regarding the suggestion of bringing diamond drill holes under the provisions of that act, who is to reimburse the operator who spends his good money for putting down those diamond drill holes? He spends his money, and if it is a question of public record it is something he wishes to keep to himself, for he is spending a great deal of money to drill and to get that information.

Mr. TAYLOR. It appears to me that records may be well left in the hands of whomsoever the State appoints to take care of those records, just as much as the affairs of our banks are published, yet the details are kept secret. I do not think there would be any serious danger arising from the records becoming public property.

Chairman FOHL. It occurs to the chair that the next subject for discussion, if there is no further discussion on this subject, is one of considerable importance and may take up quite a bit of time. It

ought to be discussed with the slides on the screen before us, and possibly its discussion might be postponed until the evening session. What is the will of the meeting?

Mr. RICE. I would suggest that it would be very desirable when you have finished with the general discussion that a committee be appointed, which I suggest might be formed from representatives of the different interests here represented; that is to say, to have two or more members from each body, the coal operators, the State inspectors, the gas-well operators, the oil-well operators, the State geologists, and from the Bureau of Mines. My further suggestion is that these different interests select their own representatives on the committee and that this committee go into session, take up and consider the different points which have been suggested at this conference, and to-morrow morning bring back a report to this conference. Then the conference could consider what should be done with the report of that committee.

Chairman FOHL. Are there any further suggestions in this connection?

Mr. WILSON. If the committee were as large as indicated in Mr. Rice's suggestion, it might include a considerable part of the people present. I would suggest a smaller committee, say, one member from each body. The suggestion of having a committee, I believe, is an admirable one.

Chairman FOHL. The chair had in mind that possibly this discussion might be concluded to-night and that such a committee might be selected to-morrow, as then there would be a fuller representation of the parties in interest. However, the matter is in your hands, and I will entertain a motion in any direction you choose to move.

A MEMBER. It seems to me that a better way to accomplish it would be to go through this proposed bill, discussing it now and finishing it, and then turn it over to a committee for report to-morrow morning. That, it seems to me, will be the better course to follow.

Mr. SCHLUEDERBERG. I think the point of appointing a committee and putting the work in the hands of this committee is all right. I think it is perfectly proper that it should be discussed fully, to give the committee points to work on. This is a great question. The subject is growing as we get into it, and it is going to keep growing. It is a matter that can not be cut off very short; therefore, it is utterly impossible to turn the matter over to a committee to-night and have the committee report to-morrow. You have got to give the committee more time, and I think the matter should be thoroughly discussed here; but it is impossible to come to any conclusion in a meeting such as this, so after discussion it ought to be delegated to a smaller committee for that work, but give that committee time.

Mr. LELAND. I agree with that, and I think the minutes of this discussion ought to be available for that committee.

Mr. SCHLUEDERBERG. I move that we proceed with the discussion.

The motion was put to vote and duly carried.

PROTECTION OF COAL BEDS.

Chairman FOHL. We will now take up the next section.

Mr. Hood thereupon read the next section relating to casing off surface water, as follows:

18. CASING OFF SURFACE WATER.

Any bore hole penetrating any workable seam of coal shall be cased by the owner of the bore hole with a suitable casing (conductor or drive pipe) so as to shut off all surface water from entering the coal seam.

Mr. HOOD. It is to be drilled to a point 20 feet below the lowest workable seam of coal that is being worked in the district, the idea being to get far enough down so that in case of any movement of the strata through mining there would be no tilting of the casing but the ground should have a good grip on it and a true seat should be furnished. The several State laws vary from 10 to 50 feet as to the distance of the seat below the seam. There seems to be no particular reason for these differences, the idea in each being that of seating the casing some distance below the coal. The idea of specifying a metal casing not less than one-quarter of an inch in thickness is to have a reasonably thick casing and yet not so thick as standard iron pipe, the thickness retarding corrosion. The space between the casing and the wall of the bore hole is made as small as we thought it possible to be to get a tight filling of clay between the casing and the surrounding bore hole. That explains the 4 inches difference in diameter. Some scheme should be adopted to keep the pipe centered in the hole so that the packing shall be on all sides of it and we can be certain that the packing extends to the surface, so that there is a complete seal between the casing and the wall of the bore hole. The second inner casing, 4 inches in diameter, shall extend 10 feet below the first casing, the difference in diameter being enough to get a good pack between the two. The extension of 10 feet below the outer casing is sufficient to make it more difficult for a leak to extend under the seat of the inner casing into the space between the two casings. The filling was specified to be of clay rather than of cement mortar because it was believed that the clay would pack tighter and in any movement of the ground would not crack but would flow and pack tightly; that if the ground moved, the stresses would not be localized and would not tend to

cause local shear or break, but a gradual bend in the pipes. These are the ideas intended to be brought out by this wording.

Chairman FOHL. Are there any remarks on this section?

Mr. CROCKER. In regard to the diameter of the casing being 4 inches less than that of the bore hole, we are drilling places where the Pittsburgh bed is 1,000 feet deep. We put in our 10-inch casing there. Suppose there were a lower bed 200, 300, or 400 feet under that. The proposed wording calls for another casing 4 inches less in diameter. In order to complete a well taking 1,000 feet of 10-inch casing, we would have to start with 16 or 18 inch casing. If the well is properly plugged, such precaution is useless. The expense would be such that we could not stand it.

Mr. HOOD. Is that lower coal being worked in that neighborhood?

Mr. CROCKER. No; but it might be 10 or 15 or 50 years hence. Probably we shall be away from there long before it is worked; but if the proposed law is placed on the statutes, we are expected to obey it. Where a well is properly plugged I do not believe there is any danger.

Mr. HOOD. The only way you can avoid that is to leave the determination of what is a workable seam of coal in the hands of some person who would be reasonable about it. For the specific case which was mentioned I do not know whether any State geologist would say that that was a workable seam of coal within the intent of this act. If not, the driller would not have to put down a 16-inch hole to that great depth. If, however, there is a workable bed of coal, some protection ought to be given. When they work that bed 1,000 feet or more in depth, then these unfortunate beings many years hence who may have to drill holes while the seam is being worked may be required to make some such arrangement as is called for.

Mr. CROCKER. If the requirement is going to be put in the form of legislation, ought not the term "workable bed of coal" to be defined, so as to make the law definite?

Mr. HOOD. We thought we would make the State geologist the definer of a workable seam of coal in any case.

Mr. CUMMINGS. Isn't that putting it into the power of the State geologist to make or destroy millions of dollars worth of property in this State?

Mr. MOLYNEAUX. If the coal seam is what we call a workable seam to-day, the Pittsburgh coal, which ranges from 5 to 8 feet in thickness, is the only bed that is worked in this vicinity; but on the other side of the Allegheny River and a little farther north the Freeport seam, which is not nearly so thick, is being worked. We know that the Freeport seam underlies the Pittsburgh, and there are other workable seams underneath; that is, they are of workable thickness, which is largely the intent of the wording here. Whether these

lower beds will be worked within the life of the gas or oil field, within this district, might be another proposition. I think it would not be well to have the act cover those lower deposits, which will probably not be worked for 50 or 75 years, when the oil and gas in the vicinity will be completely exhausted. If those measures are worked at that time, the records will show where these wells are, and proper precaution can be taken. I think it would be a serious burden on the driller to have to case down to those lower depths at this time, where the probability is largely against the coal being in the market in the lifetime of the oil or gas.

Mr. RICE. I think any committee appointed will be glad to have a discussion of this point, which was one of the most difficult things we had to consider, because we realized that what is commercially workable to-morrow might not be commercially workable to-day. Southwest of Pittsburgh, where the Pittsburgh coal is deeper and may be underlain by the Freeport and other seams, it is not probable that within this generation, or perhaps the next one, the Freeport coal will be worked, even if it is of workable thickness. Yet we can not tell when deep mines may be started, I admit.

Mr. LAYTON. I should like to know how you can puddle that clay down the outside and between the casings.

Mr. HOOD. I would rather have Mr. Heggem answer that question.

Mr. HEGGEM. Clay can be mixed with water so that the water will carry about 40 per cent of solids, and this fluid mixture will flow into small crevices. It can be put in as fluid, either through another pipe lowered to the bottom, or by means of a bailer, or by being poured. As regards pouring from the top, there is some doubt as to whether the clay will bridge, but it can be poured down through a smaller tube. A provision is made in these suggestions that the first string of casing need not be set in the cement, whereas the second string of casing is set in the cement. The cement does not come up to the shoe of the first string of casing. That permits the second or inner string of casing to be put in first, and gives a larger space, not less than 4 inches, on each side of the second casing in which to introduce a 2-inch tube and force the clay down. After the clay is put in it will settle. It may be allowed to settle immediately, or it may be kept in suspension for a longer or shorter period. While it is still soft the first or outer casing can be put in.

Mr. HICE. What will you do in case of striking a flow of water? The water will probably rise to the top, or the clay will not settle through the water.

Mr. HEGGEM. You can make the clay settle.

Mr. HOOD. That question was asked of a clay expert We had a little doubt as to how that clay would act, and we asked him this question: "If a clay is puddled and introduced into a pipe or

a space like this, how long will it take to settle? Is it a matter of minutes, or days, or months, or years?" He replied: "You can make the time anything you want; if the solution is accidulated precipitation is rapid, if the solution is alkaline the clay can be held in suspension for a considerable time." It is a matter of manipulation. The amount of acid required, the degree of acidulation, is very mild indeed. It takes only a little acid to settle the clay. Dr. Bleininger, of the Bureau of Standards, is my authority.

Mr. HICE. I agree with him, but under the peculiar conditions, with quite a number of years' experience in the clay business, I do not think that clay so settled would have sufficient density to accomplish the desired result, but think it would largely adhere to the sides or the walls of the pipes, in the way clay does in casting ware.

Mr. HOOD. Would that be true with any considerable hydraulic pressure?

Mr. HICE. Undoubtedly so. Clay wares are cast under pressure.

Mr. HALL. What you get would be flocculence. Of course, the probability is that the water down there is acid with hydrochloric acid from the well, but I do not think you would get much more than flocculence.

Mr. HICE. Clay so settled would be very different from puddled clay. It would be more like soft slime.

Mr. HOOD. I would like to ask Mr. Heggem whether this method is being followed in any case that he can think of; that is, the use of clay for filling holes such as this?

Mr. HEGGEM. Clay is being used in drilling oil wells chiefly in Louisiana, Texas, and California. It is used in various degrees of fluidity. Perhaps I might cite Louisiana, where oil is found below the gas. A driller may strike gas at 750 or 800 pounds pressure. He has safety appliances to control that gas, and on top of the inner casing he mounts a gate valve and puts about 40 feet of pipe above that valve, and another gate valve on top of that. Then he closes the lower valve, fills between the valves with thick mud or clay, closes the upper valve and opens the lower valve, and the clay slides down into the well. This operation is repeated until the pressure of gas is entirely removed. Muddy water is pumped down then to finish the job; the driller then proceeds with his drilling, and is not bothered with gas coming in. As to clay being used in the way we have suggested, I do not know of a case. We are simply making the suggestion.

Mr. DE WOLF. Would sand pumpings fill the bill just as well?

Mr. CAMERON. I think it would, rather better. I believe it is the same with cement. In cementing our casing through the coal, we are drilling our holes to the proper depth, filling the hole until we get

above the coal at the proper depth, and then are forcing the casing down to the bottom of the well, bailing the cement out of the casing and pouring it down the outside. In this way you know that the cement is in the proper place, whereas if you pour it in first, you do not know where it stops.

Mr. HOOD. Your preference then would be to put in cement; that is, 1 to 2 mortar?

Mr. CAMERON. Yes; and then fill in between the casing and the side of the hole; for the latter purpose sand pumpings would probably be just as good. Sand pumpings get as hard as slate.

Mr. CORRIN. In several localities you get gas at four or five different levels, and we may be using three or four different pipes. If you have two casings to start with, you might have five or six sands to reach, carrying different pressures of gas. Start a hole with 18-inch, and you never in the world could get down to the lower sand. We are cementing wells now about 990 feet deep, and put 10-inch pipe in, and I know we could not put in larger sized pipe at all.

Mr. HOOD. It may be that the gentlemen have other suggestions; if so, suggestions are exactly what we want. If any one who does not care to discuss it now will send a sketch or written memorandum later, we shall certainly be glad to have it.

Mr. RICE. We might invite Dr. Bleininger to be here. He has had very wide experience with clays and is regarded, I think, as an authority on clay working.

Mr. HICE. There is no question as regards settling by acidulation.

Mr. TAYLOR. It may be a matter of interest here to read to you part of an agreement that we had with a number of gas companies in the way of plugging and cementing wells, which they have agreed to do. When a well is abandoned the 2-inch pipe from below the coal bed to the surface is for a vent, so that there will be no pressure upon the coal. We have found trouble in complying with the requirements in casing a hole, or at least in getting what we considered good work. Frequently the space between the casing and the walls of the well is filled with water, and if the cement is poured in at the top the cement and sand separate, and there is only a sand filling at the bottom. If a pipe about 1-inch or 1¼-inch were inserted outside of the casing, the cement can be forced with compressed air clear to the bottom, and the pipe can be lifted and the hole filled in a perfectly satisfactory manner. We would be entirely safe if the cementing were done in that way, since the space between the rock and the casing would be thoroughly cemented. This takes care of only the Pittsburgh coal. To carry such cementing to a greater depth would be a hardship on the oil-well people, one that I do not see how you could overcome. To carry out the method to 1,000 feet or 1,200 feet, I think, would be impracticable.

Mr. LAYTON. There is no great objection to this part of his agreement, but there is another section immediately following that that might be of interest. I wish you would read that.

(Mr. Taylor read further in the contract.)

Mr. LAYTON. The only objection we had to this agreement was just a little rider, as I call it, that we were to save them harmless from all water and everything else forever, in addition to the features mentioned. There is one thing we want, speaking for myself individually, and that is a reasonable method. We want a method for plugging abandoned wells under a just and fair inspection. Now, is it possible to eliminate some of the details mentioned and put the matter in the hands of a disinterested department that would give us proper results?

Mr. DE WOLF. In connection with committee work I wonder whether we have proceeded far enough in the reading of this proposition so that such a committee might find it convenient to begin its work this evening.

Mr. RICE. I think we should have a committee which should report to-morrow, although I believe there is much to be discussed.

Mr. HICE. If anything is to be done it ought to be whipped into shape so that it could be presented to the legislatures promptly. Some of the legislatures in session now will adjourn early in March.

Mr. HOOD. It will be better to have it right than to hurry it.

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Mr. RICE. We should try to get everybody out to-morrow morning, and to then appoint a committee which might proceed to-morrow afternoon to lay the framework or at least arrange an organization, so that those who could not attend might present a written discussion. The committee might then assemble at a later date and report back to the adjourned body.

Mr. SCHLUEDERBERG. I would like, as Mr. Rice suggests, to hurry this along, but it is too big a question to rush. We have got entirely too much to do. There are too many interests to consider. We have to consider not only the coal interests but the oil and gas interests and numbers of others. We have been working on this thing with our own gas wells in connection with coal mines for the last 20 or 25 years. Things are going on that are not exactly right, and we can not make them right in a week. This is not a matter of days; it is a matter perhaps of weeks.

On motion of Mr. Schluederberg, adjournment was had until 10 o'clock Saturday morning.

The Saturday morning session was called to order by Chairman Fohl, who stated that one of the gas companies had handed in a plan, showing a method of protecting a well, and asked Mr. Corrin to make what explanation was necessary in connection with slides

thrown on the screen. [The plan showed a bore hole through a coal bed to a series of gas sands.]

Mr. CORRIN. There is not very much explanation. It gives the plan of sinking a 16-inch hole 15 feet below the bottom of the Pittsburgh seam of coal, and then putting a shoe at the bottom of the 10-inch casing and inserting the 10-inch casing. The shoe is to keep the 10-inch casing in the center of the hole, so that the cement will be equally distributed around the casing. Before the cement is put in, the hole is drilled 100 feet below the bottom of the Pittsburgh coal. The idea is that if you put the cement in first and drill through, you will break the shoulder off. If you drill the 10-inch hole through before cementing, and put in thin concrete, there will be no danger of breaking it at the bottom. Then cement is poured in to about 50 feet above the top of the Pittsburgh coal, and allowed to stand, and afterward sand pumpings are put on top of the cement to the surface. This explanation is practically shown on the next slide.

Mr. RICE. May I ask if the space between the inner and outer casings has a vent to the outside?

Mr. CORRIN. That is where we generally take out the gas coming from the upper sand. We sometimes get a little gas between the 10-inch casing and the hole, but generally there is a vent into the low-pressure line.

Mr. RICE. The shoe may be under some small pressure?

Mr. CORRIN. Yes.

Mr. RICE. Do you have an automatic relief valve, that is, one that will lift at a certain pressure? Suppose there is a sudden outburst of any sort, how would you keep that gas from forcing its way up under the shoe?

Mr. CORRIN. The low-pressure line would take care of that, but there is not a complete vent straight through; that never could be. The low-pressure line is one that has little or no pressure on it.

Mr. FOHL. Is there not a vent between the 10-inch and the next pipe always?

Mr. CORRIN. Yes, sir; practically.

Mr. HOOD. There is a low pressure in there?

Mr. CORRIN. There would be only 1 or 2 pounds.

Mr. RICE. Doesn't it give the opportunity, though, for some man, not intentionally, to close the vent, so that there will not be a vent to the outside?

Mr. CORRIN. That did happen once.

Mr. RICE. That would be an objection to that particular system.

Mr. MOORE. Don't you usually take the gas up in another pipe inside of that 10-inch?

Mr. CORRIN. Yes; there is a 10-inch casing put in and inside of that an 8¼-inch and inside of that a 6½-inch and the high-pressure gas comes then in a 5-inch pipe inside of that.

Mr. MOORE. Do you take gas up in the 10-inch casing?

Mr. CORRIN. Yes; that would be merely surface gas.

Mr. DE WOLF. What is the extreme depth you carried this method?

Mr. CORRIN. The deepest I know is 990 feet.

Mr. DE WOLF. Is that handled by a 16-inch hole all the way?

Mr. CORRIN. A 16-inch hole clear down to 50 feet below the coal. The high-pressure gas always comes between the 5-inch and the tubing. The only gas that would come up in the 10-inch would be the shallow gas with a very low pressure.

The second slide contains the following written explanation:

"Drill a 16-inch hole 50 feet below the coal. Then stop and insert a 10-inch casing with a shoe, or a swedge nipple, at the bottom, in order that the casing may be certain to fit in the center of the hole. Before doing anything more, or doing any cementing, drill a smaller hole 100 feet below the point where the 10-inch casing rests. Then insert cement on the outside of the 10-inch casing so that it will come to 50 feet above the top of the Pittsburgh coal; then after the cement has set fill to the surface of the ground with sand pumpings; after the 10-inch casing has been inserted, and before starting to drill the hundred feet above mentioned, provide some protection for the space between the 10-inch casing and the wall of the 16-inch hole, so that no sand pumpings or other foreign matter will get into it before the cement has been inserted. The idea of drilling a smaller hole 100 feet deeper, after the 10-inch casing has been inserted, is to prevent any breaking of the cement by the hammer of tools on the casing in drilling."

Mr. HOOD. Why is it 50 feet above instead of any other amount?

Mr. CORRIN. It is merely a matter of judgment.

Mr. HOOD. I should like to know whether any casing thus protected has ever been uncovered in coal mining, so that anyone knows in what shape that 3-inch coating of cement is, whether it is really intact.

Mr. CORRIN. We insert the cement through a 2-inch pipe put down to the bottom of the hole and worked around the casing. As it fills the casing the pipe is gradually raised. The idea is to try to distribute that cement evenly around the casing.

Mr. HOOD. Those who have worked with considerable quantities of cement know that it sometimes swells and sometimes shrinks, and you are not always sure just what does happen, and it is rather difficult to get a job of cement that has a perfectly clean face. In this case it seems to me it is questionable as to whether the cement

really gets in and whether the cement is water-tight. Personally I am inclined to be doubtful.

Mr. WATTERS. In regard to that shoe, how much larger in diameter would it be than the collar of the 10-inch casing, or what space would there be between the casing and the shoe, and might the cement get away from you at that place?

Mr. CORRIN. The shoe is intended to be just large enough to go down in the hole; to have the same diameter as the hole.

Mr. WATTERS. How about the collar passing the casing? The collar would have to pass through this shoe.

Mr. CORRIN. The shoe is on the outside of the casing.

Chairman FOHL. Are there any further questions? If not, we will proceed with the discussion of the paper.

Mr. DE WOLF. May I ask a question of the last speaker? I suppose, of course, there would be some economy if, on abandonment of the well, it would be practicable to cut that 10-inch casing 50 or 60 feet above the coal vein and pull it out, or is it necessary to leave it all in? In Illinois we have always left it in.

Mr. CORRIN. I do not see why it could not be cut with a right and left hand thread and screwed off above the coal.

Mr. Hood then read the paragraph entitled "Casing Through any Coal Seam," as follows:

(19) CASING THROUGH ANY COAL SEAM.

Any bore hole drilled for gas, oil, or other mineral shall be drilled to a point at least 20 feet below any workable seam of coal that may be penetrated and receive a metal casing not less than ¼ inch in thickness and of an inside diameter 4 inches less than the diameter of the bore hole. This casing shall be concentrically seated on the bottom of the hole, shall extend to the surface, and shall be known as the first casing.

A second inner casing 4 inches less in diameter shall extend at least 10 feet below the first casing and shall be seated in 9 feet of cement mortar composed of one part Portland cement and two parts sand. The inside of the second casing shall be open to the atmosphere its full length.

The intervening spaces between the second and the first casings and between the first casing and the bore hole or outer wall shall be filled with puddled clay to a height of 30 feet above the coal seam.

The idea of this is that a central casing going through and accessible from the excavation might be rapidly corroded at the floor, where water would flow. It might get through into the 2-inch space inside left for the clay; so to protect the outside casing from corrosion a wall is put up, with a 2-inch space between that wall and the casing, to be filled with puddled clay. In order that this may not act as a mine pillar in any place where it is not wanted, the inclosing wall is made small.

Chairman FOHL. Are there any remarks or discussion?

Mr. HALL. I do not believe that the discussion of the previous subject was really finished. The main feature of the suggestion of the Bureau of Mines, outside of that of having a vent inside the outer or first casing, is that the use of clay between the casing and the sides of the bore hole provides a certain elasticity or flexibility. I think the point is well taken. I think that what the coal operator has demanded in the past has been a perfectly rigid hole, which is the very thing he does not want. When we are dealing with masses of rock, the weight of which may run anywhere from 4,000,000 to 9,000,000 tons, such rigidity as the operator has been demanding and getting by a few paltry inches of concrete and a small iron pipe is wholly inadequate. And whether that proposition of the Bureau of Mines does really give sufficient flexibility is open, I think, to question, in view of the fact that it proposes that there shall be a difference between the diameters of the two pipes of only 4 inches. That means, you see, that there will be a difference of only 2 inches on either side, but as the couplings will take up three-fourths of an inch on each side, the size of the couplings varies a great deal with the kind of casing, thus reducing the thickness of clay to 1¼ inches. That seems to me to be wholly inadequate, and it seems to me that it is really necessary to consider what does happen when any movement of the strata takes place. When a body of rock rotates about the edge of the coal pillar, say 15 feet long, * * * there must be some considerable sheering at certain specific points. The question is whether it would be possible to give a certain degree of flexibility. My suggestion is to drill a hole somewhat larger than was suggested, and to lower into it a spiral riveted pipe of a diameter possibly 2 inches less than that of the hole. That spiral riveted pipe would not cost much and would not last long, but it is not intended for anything more than a form. This should be filled up with concrete or cement mortar in such shape that it would be possible afterwards to put in clay grouting; then put in, say, an 8-inch tube, at least less than the diameter of this outer pipe; the clay not to be put in puddled, but in an intermediate size. The size would be a matter of determination, but we will say up to No. 3 buckwheat, without large pieces and no excessively fine stuff that would not settle. Then there would be a chance for the whole pipe to move somewhat. Otherwise, with only an inch and a quarter, as now provided by the bureau's suggestion, there might not be sufficient flexibility.

Several other things connected with the breaking of measures are interesting. If, for instance, you put a beam across two piers and hang a pipe from it, as soon as the beam begins to bend it lowers the pipe down into the opening, and there may be considerable strain on the pipe if it is not properly centered in the hole, but if it is hung in the center there will be hardly any strain. I think the important

suggestion made by the Bureau of Mines is that we should have a clay face so that the pipe can slide. Anything that is rigid is the worst thing that the coal operator could ask for, because the barrier he is trying to put up is totally inadequate for the purpose for which it is designed.

Mr. MOORE. I think there is one feature that Mr. Hall has overlooked—that is, the possibility of a lateral movement in the strata over the coal which, as any mining man knows, is of frequent occurrence; not necessarily a general squeeze of a section of a mine, but you haven't any assurance that an entire piece of roof 150 feet long will fall down vertically. It may not be 150 feet, it may be 80 feet long, and when it drops it is just as liable to slide a little to one side as to fall straight down. I do not subscribe to the theory of taking coal out around that hole. I may be a standpatter, but I am in favor of a rigid support around the casing for the simple reason that a squeeze is one of the meanest things you have to contend with in a coal mine, and no matter how good your mining practice if you get any lateral motion whatever, as you may have with all the coal out around that well, it will cut any number of strings of casings or any amount of cement or clay that you might put in there, as a pair of shears clips a string.

Mr. CROCKER. Why is the clay put in? If you want movement why not leave it out entirely? The clay stuffs the hole to a certain extent.

Mr. HOOD. Relative to these questions of horizontal shear that may be applied to such pipe as that mentioned, it is very easy to get all sorts of stresses sufficient to cut a pipe right in two. We have been looking for some evidence that that has ever happened and we have not been able to find it. Those who make a practice of testing materials know that it is very difficult to get a true shear of any material. Two planes, sliding one on the other, must come close together. What we actually get is a bending with a short lever, because that is what occurs in most shears. In the case of a rock shear, such as we are considering, it is just a question of how long that lever is. Maybe that lever will be increased by a clay pack. It means that the stress will be distributed around the tube that is to be sheared, when if there is nothing in there at all the two planes might come closer together and make a shorter bearing. That is one of the objects of the clay pack—to distribute the stresses.

Mr. LAYTON. Your fulcrum would be pretty small wouldn't it if there was a split 6 feet long and 2 or 3 feet at the back?

Mr. HOOD. Very small indeed, too small. I think if we ever had a case such as Mr. Hall mentions nothing a man could put in there would stop it.

Chairman FOHL. Is there any further suggestion along this line? If not we will take up the subject of casing through accessible mine excavation.[a] Are there any remarks on that, as to the method of protecting these pipes coming down through the workings? If not, the next matter is casing through inaccessible mine excavations. Will you read that?

Mr. Hood read the section indicated, as follows:

(21) CASING THROUGH INACCESSIBLE MINE EXCAVATION.

Where a bore hole passes through an inaccessible mine excavation the outer casing shall be protected by cement mortar held within a metal tube with a diameter 4 inches greater than the diameter of casing and extending from 4 feet below the mine floor to 4 feet above the mine roof or cave.

Mr. HOOD. There is perhaps a little inconsistency here, in that we have asked for cement in this place, and clay in the other, and we recognize the inconsistency, and perhaps can not put up a very good argument for the cement in this place, except a seeming necessity of keeping this distance as small as possible. We recognize that to make the hole larger there would require the use of some sort of a form put down through this hole, and we have not even asked for a casing; we have asked for a metal tube. It might be a spiral pipe or it might be anything that would hold the cement while it is setting. We will get it in as best we can. It was thought that cement would protect the outer casing that we provide, or the first casing, from acid mine water. If clay were put in, just as soon as the outer metal tube had rusted away, the clay would slump and go away, and there would be no protection at all, and that is the reason for putting in cement rather than clay in this place. Yet it is recognized that the cement will crack, it will not be thoroughly waterproof, and acid will in time go through it.

I may say that this matter of corrosion is a function of the rapidity with which the water is changed. If water could be held absolutely quiet and still, there being no replacement at all, or very, very slow replacement; the corrosion would be correspondingly slow. It is not believed that this cement will be an absolute protection, but it will simply be a retarder to the flow of fluids, and it

[a] (20) CASING THROUGH ACCESSIBLE MINE EXCAVATION.

Any casing that passes through a mine excavation shall be protected by a wall of concrete or of masonry or brick laid in cement mortar, extending from 2 feet below the mine floor to the mine roof. Between the first casing and the wall thus constructed there shall be left an annular space of not less than 2 inches which must be filled with puddled clay. This work shall be done by the well owner.

Casings that are exposed by mining operations shall be covered by the mine operator in the manner prescribed above.

will, therefore, retard the possible corrosion of the outer casing, which is provided for.

Mr. JOHNSTON. I have been interested in bringing forth a system of cementing that is used in shaft working for shutting off water. In a shaft that we sunk last spring for the Pittsburgh Coal Co. we ran into a seam of gas that forced the water out of the drill holes that we were drilling. We applied this system and shut off that gas and water completely. In speaking with two or three mining men the subject of oil-well holes and gas-well holes was brought up, and I suggested this system of grouting. I have listened to the previous speakers, who mentioned forcing the cement into the bore hole. It is perfectly possible to force this cement under pressure into the crevices of the rock until the rock is absolutely impervious. It is possible to make a cement lining for the well that is absolutely waterproof, and this must be done by pressure by the use of a plug on top. I am not familiar with oil wells and gas wells or coal mines except as regards the shafts going into mines, but this system is so successful that it ought to work. It was used in the Catskill Aqueduct. I believe that men who are familiar with the actual details of drilling holes and plugging them could find some application of this process.

Chairman FOHL. Could you give us a brief description of your method of application?

Mr. JOHNSTON. Putting in cement between a spiral riveted pipe and casing was mentioned previously. That would be put in to make the lining waterproof, but the cement grouting would have air bubbles in it, and, of course, air bubbles would make it pervious after the casing corroded. If the grouting were put in under pressure the bubbles would come out.

Chairman FOHL. The cement, if put in as they describe, would carry air bubbles?

Mr. JOHNSTON. Yes, sir; be full of air bubbles.

Chairman FOHL. I think your explanation of how this is done would be very interesting.

Mr. JOHNSTON. The scheme as we use it is for plugging holes in drilling rocks at any depth. Suppose we are sinking a shaft. The ordinary method would be first to drill a hole in advance, to make a sump, and then to put in side holes. If we get water in the first hole, we then drill three or four more holes. We then plug all the holes we have drilled, using tight swedge nipples with valves on top. Each nipple is packed in with bagging, or a gunny sack, or a rim of cement, or anything to make the hole tight. The swedge nipple, when driven, will be absolutely tight. We connect with this a grouting machine which was patented two or three years ago and has been very successful.

It is nothing more nor less than a tank in which the cement grouting has been put very thin, and then the top of the tank closed and connected with air pipes worked with compressed air. The cement is mixed by this air. It is run back and forth, so that it is fairly mixed. The air pipe is attached to the tank, and the compressed air drives this cement out through the pipe into the hole and completely closes it. We have got as high as 335 pounds pressure on some of those holes and have absolutely plugged the rock. After all the holes have been plugged we drill another hole to see if the strata still leak. We have seen crevices that were completely filled when we got through. This process, to my notion, could be used in as deep a hole as you want. Suppose you had a hole 300 or 400 feet deep, and you wanted to fill it so completely full that gas would be kept from getting up into the mine. Put in a plug, put the pipe in the plug, and force in cement until the hole is absolutely plugged and the crevices all around the plug are full. It would be perfectly posssible to drill through a body of rock without having the water or the gas leak out after you had drilled through. We have taken shafts—I mention shafts because I am more familiar with them—and have made them absolutely waterproof by this process. I certainly believe that there is some application that can be made with reference to oil and gas wells.

Mr. LAYTON. Anyone liking to see a machine of this kind can do so by visiting the Mount Washington Tunnel, where the same thing is being used now. They had trouble, and they have been working for a year and a half trying to fill in back of the cribbing of the bridge of the arch. It is the very same thing the gentleman speaks of here now.

Mr. RICE. This system of filling porous ground with cement grouting is certainly a most excellent one in shaft sinking, and it may have some application here. In 1908, in the Pas-de-Calais coal field of France, I found they had been using for several years a system of cementing the fissured chalk measures which must be passed through in shaft sinking. The chalk measures are so filled with water that it was formerly necessary to employ the freezing process. In 1911 I found the cementation process, as it is there called, had come in almost to the exclusion of the freezing process in that district. The hole is drilled in a circle around the projected shaft in much the same manner as for the freezing process; then cement grouting under high pressure is pumped into the hole, successively filling the fissures and joints at different horizons. As a result what is practically a concrete pillar is built. The shaft is sunk through this as in dry rock.

Chairman FOHL. Are there any further remarks on the immediate subject of casing through inaccessible mining excavations?

Mr. HICE. I should like to ask how you arrive at the designation of 4 feet above the mine roof—whether that would be sufficient?

Mr. HOOD. Just the same way that other persons have arrived at the 10 or the 50 feet.

Mr. HICE. It would not be above immediate falls.

Mr. HOOD. I believe a suggestion was made that in the ordinary coal mine it should not be enough to be more than one length of pipe. With an 8-foot seam, 4 feet above and 4 feet below, would make 16 feet. If we call for 10 feet above, as has been suggested by Mr. Rice in a preliminary conference, you would have more than a pipe length, which would require a joint; but we haven't any real good reason for a specific distance.

Mr. HICE. If it were 4 feet below the coal, and you have an 8-foot seam, with 12 feet, you will have most of this down, so that with a 4-foot fall your pipe would be open.

Mr. HOOD. And if we tried to make it long enough to take care of that situation there is no telling how long it would have to be. Is that a situation that can be met? To go a little further on Mr. Hice's line of thought regarding this cement protective layer, the useful portion I consider to be the bottom 2 or 3 feet, or as high as the water level is likely to rise; above that it is of little use.

Mr. HICE. Another question would be the matter of getting in the cement properly.

Mr. HOOD. Yes. And while I am on my feet I want to call attention to one point. I do not think the gentleman in referring to his method of puting in grout under pressure means to imply that the air is taken out of that cement. It is there somewhere.

Mr. JOHNSTON. There is usually a vent pipe.

Mr. HOOD. But I can't see how the air would get out with the arrangement described.

Mr. JOHNSTON. Through the vent pipe.

Mr. HOOD. What actually happens is, I think, that under your high pressure your little bubble of air is compressed to a smaller diameter. The cement is still porous to the extent of having those openings in it, but they are so much smaller that the surface tension of the liquid is enough to keep it in instead of allowing it to flow through. If we could get that pressure in any other way—for instance, if we had a hole 600 feet deep filled with water—we would get exactly the same condition as with mechanically produced hydraulic pressure, and the same little air bubbles would be just as much smaller.

Mr. LELAND. If it is not feasible to make this arrangement waterproof, it would still be of some use, if we had a large movement of the roof, as large as was suggested by Mr. Hall.

Chairman FOHL. The point is well taken. Are there any further remarks on this subject? If not, we will prceed to the subject of casing to exclude water.

Mr. Hood read the paragraph referred to, as follows:

22. CASING TO EXCLUDE WATER.

Before a bore hole is drilled into a gas-bearing or oil-bearing formation a string of casing shall be so set as to exclude all water from the lower bore hole.

Mr. CORRIN. I know a section in West Virginia where there are 11 different levels from which you get gas. How would you take care of it?

Mr. HOOD. I do not see how it would be possible to maintain a vent to the surface so as to keep gas from escaping through under the casing seats, if the well is not cased for each one of those levels. Perhaps we are trying to ask one well to do too many things.

Chairman FOHL. I should like to ask whether this coincides with present practice in the matter of taking gas from wells.

Mr. MCCLOY. It does not. We take gas on the inside of the tubing, and also on the outside of the tubing. We have 6⅝-inch, 8¼-inch, and 10-inch. We take gas out on the inside of the tubing, and also between the tubing and the casing. That is absolutely necessary in most fields in West Virginia.

Chairman FOHL. Are there any further remarks? If not, we will proceed to tubing gas and oil wells.

Mr. Hood read the paragraphs referred to, as follows:

(23) TUBING A GAS WELL.

To conduct gas from a gas well, tubing shall be inserted with a suitable packer placed below the inner casing and so constructed as to prevent the escape of gas except through the tubing.

The inner casing shall be left open to the atmosphere throughout its full length.

(24) TUBING AN OIL WELL.

To conduct oil from an oil well, tubing shall be inserted and extend from the oil-bearing formation to the top of the well.

The inner casing shall be left open to the atmosphere throughout its full length.

Should gas be liberated in the well in sufficient volume to have commercial value, it may be shut in by means of a packer placed below the inner casing and may be conducted from the well through tubing inserted into the well parallel to the oil tubing.[a]

Mr. CROCKER. I consider it not at all practicable to have two strings of pipe in the well within the casing. In the present practice

[a] See figure 11.

there is always the outside casing to protect from leakage into the mine, and we have wells from which we would not dare let the gas all out. It would hurt our production at once. It is very seldom that we have over 40 or 50 pounds pressure on the outer casings. If we open these to the air, we lose the gas immediately. We use it for fuel. I do not think it would be possible with a $5\frac{3}{16}$-inch hole to get two strings in unless you used very small pipe, let alone injuring the well by running it entirely open. Ordinarily we have to keep a little pressure on to keep the gas line full.

Chairman FOHL. Is it your idea to increase the diameter of the hole to accommodate your other two pipes?

Mr. HOOD. It seems to me it comes back to the question of whether we are not asking too much from one hole. It seems that in order that there shall be no leakage under the seat of the casing there must be no pressure, and as the packing with cement or clay may be not perfect, the only absolutely safe course is to keep the pressure from off those seats. As long as that pressure is on the seat and the work may not be well done, it might be that that pressure will work up under the seat, outside of the packing, and then through the ground into the mine, as it has done in some cases. It is recognized that the problem is extremely difficult.

Mr. CROCKER. But there is another string of casing outside if there is any leakage.

Mr. HOOD. That furnishes sufficient protection only as long as there is no possibility of a leak getting up under the seat that is just below the mine floor.

Mr. CROCKER. There are generally two or three strings for protection; oftentimes three.

Mr. HOOD. So that, as a matter of principle, you say the plan proposed is all right, but, as to the specific working, it is wrong?

Mr. CROCKER. Yes.

Chairman FOHL. Is it your idea that the principle he suggests coincides with the methods you suggested for passing through a coal bed?

Mr. HOOD. I think not; I am not so sure about that. I would have to study the question.

Chairman FOHL. That would be a matter for the committee to study—the details. Are there any further remarks? If not, we will proceed to the next subject, "Casing Heads."

Mr. Hood read the paragraph mentioned, as follows:

(25) CASING HEADS.

Casing heads must have at least one opening to the atmosphere to which a valve or plug is not attached, so as to insure ample vent in case of leakage into casing spaces.

Chairman FOHL. That is to give a vent or to be sure in some cases that the vent is so maintained that it is not an easy thing to deprive the well of a vent. Now, just how that should be arranged is the question.

Mr. CROCKER. The criticism that applies to the other would apply to this. The inside casing—if you left that open, you could not utilize the gas entering it. As long as the other strings of casing are in, protection would be afforded the mine.

Mr. HOOD. If the outer casing was connected with an open vent to release the pressure on that seat, it would fulfill the requirements.

Mr. CROCKER. That clause is taken care of by the other one.

Chairman FOHL. Is there any further discussion? If not, we will proceed to the paragraph "Completion of Well."

Mr. Hood read the paragraph referred to, as follows:

(26) COMPLETION OF WELL.

When a bore hole has been drilled and put into operation the owner shall file with the inspector a statement of the total depth of the hole, the sizes and lengths of casing used and remaining in the hole, the depth and thickness of all coal seams penetrated, and whether oil, gas, or water is obtained.

Chairman FOHL. Are there any remarks? If not, we will proceed to the paragraph "Suspended Operation."

Mr. Hood read the paragraph mentioned, as follows:

(27) SUSPENDED OPERATION.

When for any cause a bore hole that passes through a workable seam of coal shall cease temporarily to be operated, the inner string of casing shall be maintained open to the atmosphere. Should oil tend to rise in the well above the bottom of the first or outermost casing passing through the workable coal seam, such oil shall be pumped out by the owner and its level maintained below the bottom of such outermost casing.

Chairman FOHL. The whole intent of this is to take care of a well that an operator is letting stand and does not know whether to abandon or not, or a well that is simply being neglected. If oil accumulates and rises above the seat of the casing immediately below the floor of the mine, it might force its way through. There would be a hydraulic pressure forcing the oil into the mine, and in some way that should be prevented. It was though also that if this requirement was prescribed it would lead to a formal abandonment of the well and a formal plugging, rather than to simple neglect.

ABANDONMENT OF WELL.

Chairman FOHL. Is there any further discussion? If not, we will take up the subject, "The abandonment of the well."

Mr. Hood read the first two sections under that head, as follows:

(28) NOTICE OF INTENTION.

When any oil or gas well is to be abandoned the owner shall notify the chief inspector in writing of such intent to abandon, and shall proceed with plugging methods only after complete arrangements for inspection shall have been made and permission granted.

(29) METHOD OF PLUGGING.

Any well that is to be abandoned must be entirely filled from the bottom to the surface. The lower part of the wellhole must first be filled with sand, clay, or rock sediment to a point 5 feet above the lowest gas-bearing or oil-bearing formation. Above each formation supplying gas, oil, or water, and immediately below any workable seam of coal, there shall be a plug as hereafter described, and each well shall have not less than three such plugs whatever the formation.

The plug above mentioned shall consist of 25 feet of cement mortar made of one part Portland cement and two parts sand properly mixed and placed with a bottom-dump bailer upon 5 feet of clay resting on a seasoned wood plug 2 feet long and of the same diameter as the hole. Five feet of clay shall also be placed on top of the cement section. The space between the plugs here called for must be filled solidly with puddled clay, sand, or rock sediment, or with cement mortar or concrete. The filling shall be complete up to the bottom of a casing before such casing can be drawn; the filling shall continue length by length as the casing is withdrawn, cement, mortar, or concrete being used in those sections of the wellhole that pass through coal seams.

Mr. RICE. The thought was to minimize, as far as possible, the expense of maintaining casing. As has been suggested, there are a vast number of wells that will not be approached by mining for years to come. Therefore, if we can restore the ground to approximately its original condition and make it equally impervious, that would be the best way for the mine. Moreover, we thought that by allowing the withdrawal of the casing the oil and gas companies would be put to a minimum expense.

Mr. CROCKER. Complete plugging is a pretty expensive point with us. There are places where it is almost impossible to get anything to fill in with, even ordinary dirt. Personally, I do not think it is necessary to fill completely if you put in a plug at the proper place. We are using now a lead plug with an iron mandrel. We find it far superior to anything we have yet had. I have in mind a gas well where a wooden plug could not be used on account of the pressure; it came right up after the tools. We then used a lead plug in the well. The plug was heavy, and we could get it down and keep it there. We got it down and got something to set it with and put the mandrel on the top of it, and in a dozen revolutions there was no

leakage of any gas. It has been down now three or four years and has never shown leakage. There is no chance in the world for it to break or go to pieces, and we consider it far superior to a wooden plug.

Mr. Hood. May I ask whether, if it were anticipated that such a hole would have to be filled, it would be burdensome to save the stuff that came out of that hole, or a good share of it, in order to put it back; that is, is it available?

Mr. Crocker. The pumpings come out pretty thin and run off a good distance, and it is pretty hard to keep them where you can get them, and I do not believe they would fill the hole either. A good deal disappears somewhere.

Mr. McCloy. What is the objection to an anchored plug; that is, a plug anchored to your last string of casing? That is the manner in which it is used at the present time.

Mr. De Wolf. I do not know to what extent the geologists were responsible for this suggestion of plugging from bottom to top. The present point of view is one we had in mind. There are doubtless in some localities a number of coal seams, aside from the major seams, that you all recognize and look for, which sometime may be available. It is impracticable for your driller to run a steel tapeline to all of those, and we recognize that however careful he is he can not absolutely locate such a seam of coal. Therefore, it is impracticable to put in a sufficient number of plugs in their proper places to protect those seams of coal. So, from our point of view, the easiest way out was to suggest that the entire hole be filled with rock material, so as to avoid that trouble.

Mr. Watters. In regard to plugging wells, I am interested in the coal-mining end of it, but I do not believe in putting a burdensome load on the gas and oil operator. For my part, I am opposed to plugging a well from top to bottom. I rather believe, from the experience we have had this last summer in plugging abandoned gas wells, that it would be better to fill it with sand to the top of the upper gas-bearing sand, and leave the hole open from there up to a point 100 or 150 feet below some particular coal seam. Then place a packer, say a disk packer, with a tube not less than 2 inches in diameter leading from the packer to the surface and extending above the surface, to keep people from dropping things down it. Next cement around this tube to a point above the coal or all the way to the surface. If the cement should not protect the tube, it seems to me that the cement lining would still leave a vent large enough so that there never would be any accumulated pressure of gas. This summer we struck a well that, had it been open at the surface, I am positive never would have caught fire when we cut into it. We don't

know yet who drilled the well, and the location was not previously known to us. We dug around on the surface to find that well. We had a small drilling machine, and on opening up that well we found that it was not plugged, but simply filled to a point about 110 feet from the surface. From there down it was open for about 715 feet from the surface to a point, as we presume, above the upper gas sand where some kind of a plug had been put in. But it was a plug in name only, because it did not hold the gas. It was more likely intended to keep the water out of the gas sand. We placed a packer 150 feet below the coal and ran the tube to the surface, but as there was question as to whether there was gas coming from above the seam we placed a sleeve packer over the 2-inch tube, with a 4-inch tube leading to the surface, and those two pipes stand 8 or 10 feet in the air. There isn't a sign of gas in the mine—not enough at a time to keep an ordinary jet burning—but it was accumulated pressure in that abandoned well that caused the trouble. We are particularly in favor of an arrangement of that kind, and I believe the gentleman representing the gas and oil companies will agree with me when I say that that method is far cheaper to them and I believe more satisfactory than any other method—that is, plugging the well to the surface.

Mr. RICE. I think undoubtedly this particular case was very well taken care of, but that was looking for the present. We have to bear in mind the coal operators of the distant future, in the areas now remote or in deeper coal beds, the positions of which are not precisely determined by the driller.

Mr. DE WOLF. As pointed out, the difficulty is that we do not know precisely where these beds are located. I would not have supposed from what has been said previously in the conference that there has been this objection on the part of the gas and oil well operators, because I understood that it was a matter of agreement, with certain of the large gas companies and certain large coal operators, that they should fill the holes with concrete or cement mortar. The expense involved would seem to be much heavier than is here contemplated, because with the use of these so-called official concrete plugs there would be perhaps four or five needed. These, if used in a 10-inch bore, would represent only several cubic yards of concrete—I haven't figured the amount of filling, disregarding the amount of the hole excvated under ground, but I believe that a 10-inch hole about 3,000 feet deep would average, if filled, about 60 cubic yards. I would suppose that in the country concrete material would be expensive, so it would appear to me to be much cheaper to use only concrete plugs and fill the rest with dirt or clay.

Mr. HOOD. The total cost of material with cement at $2 a barrel and sand at $1.25 or $1.50 a yard—and I know that where you have

to haul it 15 or 20 miles $1.25 or even $1.50 is entirely too low—the total cost for material for plugging will be, say, $30 to $50.

Mr. HICE. There is another matter in putting a pug at the bottom of the well. I think we have cases where plugs have been known to move in a well, and therefore not perform their real duty for 100 feet or so. That is one of the necessities for plugging a well from the bottom up.

Mr. WATTERS. Mr. Rice's remarks bring another thing to my mind. When we had the troube I spoke of, the gas companies showed the greatest willingness to help us out, and we have never had any trouble with them in that respect. However, we were going to plug it ourselves, and the expert drillers we had from all the surrounding country advised against it, for the reason that in that particular locality the strata between the coal measures that we were working and the upper gas sand seemed to contain underground water currents. They told me that in drilling some of those wells in that locality the water came out of the well entirely clear all the time; that is, there was such a current of underground water that the water was not discolored by the pumping. Whether that is true or not, some of the gas people and geologists might tell us. In that case, I do not see how we could get enough of the liquid cement down to plug it.

Mr. HOOD. Speaking of the 2-inch vent, what is your judgment as to what would be its condition after some years, when that part of the mine in its vicinity has been worked out and abandoned, and the roof has begun to come down, and you are working in some distant part connected with these workings? Would you still feel safe with that small 2-inch pipe sticking up through this part of the mine, which you could not get at, and knowing that the roof was coming down? Do you still think it would be a safe vent?

Mr. WATTERS. I do not believe it is possible in any way to guard against the breaking of the tubing in that case, particularly where there is a heavy covering. But in ordinary cases, to my mind the 2-inch tubing would be enough.

Mr. HOOD. I think you are right, perhaps, in ninety-nine cases out of a hundred.

Mr. RICE. I would think that to plug a hole in the way now done in some coal properties would be more expensive than the plan suggested of simply filling it, except for short concrete plugs, with any material that might be available.

Mr. HALL. I believe any arrangement ought to take into consideration the distance between the coal and the gas sand. If the two are close together, of course the plugging is not sufficient, because complete plugging would raise the pressure so much that the crevices would very easily bring the gas to the coal seam. That whole matter

could be adjusted by specifying a certain distance that must exist between the coal bed and the gas sand if simple plugging is to be used.

Mr. WILSON. My thought is that there is possibly an excellent field over here for some actual experimentation, and as the conference, or the committee that we may appoint, may take some time to come to any conclusion, there may be an opportunity, perhaps, for the Bureau of Mines to cooperate with oil and gas men in actually plugging wells with various materials and ascertaining the facts by tests.

Mr. CORRIN. Answering something that was said a few minutes ago, we had a well in which the gas had a very high rock pressure, and we were for more than three months trying to plug that well. It finally cost about $5,000 to plug it. We tried the lead plug and it failed.

Mr. HICE. The diagram shown calls for two lead plugs. I simply want to ask where those lead plugs were placed, whether they were placed in shale or in sandstone. I can readily conceive of sandstone such that they could be put in and seemingly shut off the gas entirely, and yet there would be sufficient passage of the gas through the pores of the sandstone up around that plug to allow a dangerous accumulation at a higher level in time, although you would not get the effect immediately, in all probability.

Mr. CORRIN. I think it would depend a great deal on the pressure of that well.

Mr. CROCKER. I would like to ask Mr. Neill to explain about this well I had in mind,

Mr. NEILL. We had quite a lot of trouble about two wells, one of which had not been properly plugged. The cost to plug it, if I am not mistaken, was $5,000 or $6,000. We built a new rig over the well and cleaned the well out. There was very little gas, maybe 100,000 feet, but not enough to use. However, the pressure was very high. With a wooden plug we could not stop it. The wooden plugs came out with the tools when we brought them out.

We put in some tubing with a flange on top and set a lead plug on the flange. At about the third revolution the gas was shut off. We drove on the plug for 10 or 15 minutes after that and then filled in above and put in our wooden plug and filled on top of that. We put in three plugs and then got out what casing we had.

We plugged each hole, 5, 6, and 8 inch, separately, and the well has never shown a particle of gas since. I do not think it ever will. It surely can not get through that lead.

The lead plug was made with a steel mandrel on the inside. The mandrel extended down about 10 inches into the top of the plug, and it was cast in that position, about 6 inches projecting from the top of the lead.

It has been about five years, I think, since we did that job, but we did not fill it clear up. We plugged below the coal, and we put in a wooden plug above in order to keep the water from going down onto the coal, and that is the condition the well is in to-day.

Chairman FOHL. Your objection, Mr. Hice, would obtain either with the lead or the wooden plug, wouldn't it?

Mr. HICE. Certainly. It would depend altogether on the character of the sand. I can conceive of a sand where the plug would shut off the gas perfectly. On the other hand, there are plenty of sands that would allow the gas to accumulate and pass around.

Mr. LELAND. If we will provide legislation for an oil and gas inspector who is competent, we should depend on his judgment to some extent, and if we find it is necessary to use a variety of stoppings, depending on the character of the rock, wouldn't it be better, instead of formulating absolute, definite legislation, to leave the details to the inspector's judgment?

Chairman FOHL. That is a problem that should be settled by each State legislature.

Mr. HICE. There ought to be a method prescribed, some discretion, however, being allowed the inspector.

Chairman FOHL. A tentative method, at least?

Mr. HICE. A tentative method, at least.

Chairman FOHL. Is there any further discussion on this subject?

Mr. HICE. As we provide for recording the location of new wells, I think the notice of abandonment should provide for a map and record of old wells of which there is now no record. That is, we would have a record of them, the same as of the wells.

Mr. DE WOLF. I would like to ask some of the gentlemen who have raised the question of expense of filling the hole whether that applies only to those inaccessible regions where materials are scarce; and may we assume that where materials are readily available the proposed method of filling from bottom to top with clay or sand or other like débris would not be excessively expensive or perhaps be even cheaper than the customary method? Is there any mechanical difficulty in getting stuff down into the hole as the casing is pulled?

Mr. CROCKER. Sometimes a piece drops in that is a little bit large. It lodges at some place in the hole. It is solid as long as fluid does not strike it, but if any water or oil strikes it, away it goes. We are plugging very successfully without entire filling.

Mr. DE WOLF. Let me call your attention again to the condition we have in Illinois, and get your cooperation in arriving at a solution of our difficulty. We have a field where there is undoubtedly an important seam of coal, but there is no mining in the vicinity now, and the drillers are prone to say there is no coal in the hole. Perhaps they are perfectly sincere. Be that as it may, it would be in-

adequate to prescribe a method of plugging a workable coal seam when no record is kept of where or what that seam is. There would be no plug in our State under such a law. If you could require the hole to be filled up with rock material, any carelessness on the part of the driller or any intentional attempt to deceive would be offset.

Mr. CROCKER. It may be possible that there is no coal in that particular place.

Mr. DE WOLF. Diamond-drill holes show that the coal is there.

Mr. CROCKER. Assuming that such is the case, wouldn't it be safer to plug under and above the coal?

Mr. DE WOLF. The geologist doesn't know, within perhaps 50 feet, just exactly where that coal is.

Mr. CROCKER. If one knows it is within 50 feet, one can put a plug down.

Mr. DE WOLF. I say they do not know within 50 feet.

Mr. CROCKER. But if they could put a plug down, it would be much safer than filling it up.

Chairman FOHL. The next business before this conference relates to the provision concerning plugging.

Mr. RICE. I think before we adjourn we should like to have an expression of opinion from the gas and oil men about the last one or two clauses of the suggested laws and regulations.

Mr. Hood thereupon read the provision relating to the log of method of plugging:

(30) LOG OF METHOD OF PLUGGING.

When a bore hole has been plugged, an affidavit in reference thereto shall be made by the well owner, who shall state the method followed, the materials used, and the length of the sections occupied by each material. The affidavit shall be certified by and filed with the gas and oil well inspector. A duplicate of the affidavit shall be filed with the county recorder by the gas and oil well inspector.

Mr. DE WOLF. Isn't that intended to be the coal-mine inspector the second time it comes in the provision?

Mr. HOOD. The intention was to file with the chief inspector, and then have a copy go to the recorder, and a variation has been suggested that the filing with the county recorder should be by the State inspector to make it more accessible.

Capt. BARGER. It is the exception, undoubtedly, but it is nevertheless what we may and do need to do at times.

Mr. Hood thereupon read the last two paragraphs relating to the release of bond and to fees.

(31) RELEASE OF BOND.

After, and not until these several requirements have been met, any bond that may have been given by the owner shall be returned.

(32) FEES.

The question of whether fees shall be charged for license and inspection is left for further consideration.

This matter of bond and the next item are perhaps inseparable. This method is going to cost money, and the question is who should pay it, the State or the gas industry or the coal industry, the people who are protected, or who? It is not intended here to suggest who. Some larger body should suggest that. Instead of the method proposed, it is suggested that a system of fees be charged, these fees to go to the State and as they accumulate they will be sufficient to do things with—for instance, to plug wells abandoned by people who are unable to plug them or are insolvent. Then the question of whether it should be a fee system or depend on a bond given by the driller is a question that should be discussed.

Mr. CROCKER. Leave that to the committee.

Chairman FOHL. If there are no further suggestions, we will proceed to the conclusion of the discussion.

Capt. BARGER. In this matter of plugging abandoned gas wells, I have followed with marked success this method that Mr. ———— has outlined. I have used a rubber plug and a wooden mandrel, and in other cases a lead plug. Out in the district where we operate there are possibly five different producing coal seams encountered, and the cost of this plugging, if it were borne by the large corporations, possibly would not work grave harm; but if borne by the individual operator it would work serious harm. A large number of wells are drilled by people who have perhaps just about money enough to drill a well. When you come to saddling them with an added sum of $5,000 (for plugging), from which they receive no benefit whatever, it is going to do them harm, and it will possibly check the corporations in a great many cases. If you are going to fill up these wells, plug the different oil and gas sands, and keep your tools going at the prices that you have to pay for day's work to-day, the individual operator would very likely go to the almshouse. It strikes me that some simpler method might be evolved by this committee which would require much less expense. The committee should bear in mind that it is not big corporations that are doing all this development, but it is the individual oil or gas operator, who can not stand the enormous expense of this proposed method when he is obliged to abandon his well.

Mr. LAYTON. I think that the method illustrated, using plugs and filling with concrete solidly, is the exception, as mentioned by our friend over here.

ORGANIZATION OF COMMITTEE.

Mr. RICE. I understand Mr. Schluederberg, of the Pittsburgh Coal Co., now has to leave and I think perhaps others and it might be well before they leave if we could take up the matter of the organization of this committee which has been talked about and as to which we all seem to be in accord.

Chairman FOHL. We are open to suggestions as to the personnel of that committee and the proposition of what action should be taken by this conference is now before you. The drift of sentiment before you has seemed to be that we should have a committee appointed to go fully into the matter. The make-up of that committee is for you to propose. We are now open for suggestions as to the manner in which this shall be handled.

Mr. SCHLUEDERBERG. This matter has been pretty thoroughly discussed. It is a matter not only for the State of Pennsylvania, but it seems to me it is a matter for other States, and it is a question that can not be settled in a day or a week and it is going to take considerable discussion and perhaps some experimenting, and therefore I think there ought to be a good committee and a representative one. I think the coal operators, the oil men, the gas men, the geologists, the Bureau of Mines, and the mine inspectors should be represented on that committee and the officers of this meeting ex officio, and I would make a motion that there be a committee appointed, of, say three of each—that is, three coal operators, three oil men, three gas men, three State geologists, three State mine inspectors, and three representatives of the Bureau of Mines—to whom this question shall be referred, to take it up thoroughly in such manner as they may see fit and proper and report at a future meeting such as this.

The motion was seconded by Mr. Rice.

Mr. DEWOLF. What would be the total membership of the proposed committee?

Mr. SCHLUEDERBERG. I have made this committee rather large, because on such committees are many who are not always able to attend, and by making it large enough we shall always have somebody present. I think there will be three from each of the departments I have suggested, and the two officers of this meeting ex officio, which will make 20 members.

Chairman FOHL. Are there any other remarks?

Mr. RICE. It may seem at first that this committee is rather large, and I had in mind two men of each, but at the same time even two from each body would make a large committee, and it may be that the brunt of the work will fall on a subcommittee whom this

general committee may appoint. The subject under discussion is a national matter, and not simply a local matter, and therefore the representation should be as wide as possible, and it seems to me, as Mr. Schluederberg suggests, that each party should, perhaps, appoint its own committee, so as to have it tolerably representative of each body. Some of the work of the committee will have to be carried on by correspondence, of course, but the appointments should as far as possible be made representative in a national, rather than local sense.

Chairman FOHL. The resolution is silent as to the manner of appointment, and that will have to be taken up immediately after we dispose of the resolution. Are there any further remarks on the resolution?

There being no further remarks the resolution was put to vote; it was carried unanimously.

Mr. WILSON. About the method of appointment it must be in order to offer a motion, and in order to give opportunity for the discussion I move that each of the interests represented in this motion that has just been passed select its own committee, bearing in mind that the distribution should be, as far as practicable, geographical, and bearing in mind also that that geographical relation concerns the places where oil and gas and coal come together.

Mr. Wilson's motion was unanimously carried.

Chairman FOHL. I should think there should be some time limit set or some arrangement made as to when these appointments are to be made, and that that should be decided right at this meeting.

Mr. RICE. I think it should be borne in mind that it will be desirable to have the work expedited as much as possible, but we want due consideration given to the subject. As to just how that would be worked out I should like to hear from the other members.

Chairman FOHL. The chairman would suggest that there be a date set now, at which time nominations should be made, and that would practically call the committee together.

Mr. HOOD. I would move that a meeting of this general committee be called for February 18. That allows 10 days for correspondence and getting these committees together. That seems to be ample.

Mr. LAYTON. I know of a meeting of gas and oil well interests to be held on the 17th and 18th in Columbus, Ohio, and some of us might be at that meeting and get on that committee.

A MEMBER. Before the date is set I would state that there will be a meeting of the oil men of Oklahoma, Louisiana, and Texas at the end of this month at Austin, Tex. It might not be desirable to set the date to interfere with that.

Mr. HOOD. I will change my date to the 19th.

Mr. WILSON. Why not make that the 1st of March? I think that the 19th is too soon. The State legislatures that are now in session will not have a chance to act on this, and they could have something formulated that is worth while before the next legislative session, so I should think we should have time enough. Therefore, I move that February 26 be substituted for February 19.

Mr. LELAND. I will second the motion, and I would like to offer a further amendment that the chairman of this meeting have authority to complete these committees if at that date the membership of any of them is lacking.

Mr. RICE. I second Mr. Leland's amendment.

Chairman FOHL. It is moved as an amendment to the amendment that the chairman fill out vacancies on committees, if such vacancies exist at the date of meeting.

The amendment was carried by unanimous vote.

Chairman FOHL. The next question is on the amendment changing the date of meeting from the 19th to the 26th of February.

This amendment was carried by unanimous vote.

Chairman FOHL. We come now to the original motion as amended, that a meeting of this committee be called for the 26th of February, and that the chairman be authorized to fill any vacancies that may then exist.

The motion was carried by unanimous vote.

Mr. LELAND. Will you kindly explain where this committee is going to meet? There was nothing said about that, and it is rather indefinite.

Chairman FOHL. I think that is for the conference itself to appoint.

Mr. RICE. It seems to me eminently proper that, as the chairman of this conference, who would be chairman ex officio of the committee, and the secretary reside here in Pittsburgh, and as the oil and gas well problem is most acute in the vicinity of Pittsburgh, this city would be a proper place of meeting, and I therefore make this motion, that Pittsburgh be the headquarters of this committee.

Chairman FOHL. Presumably this room will be the place of meeting, so that will be attended to later.

Mr. MOORE. I would like to ask how the members of this committee are to be appointed by the respective interests.

Mr. HOOD. They will have to get together. It is up to you.

Chairman FOHL. It seems to the chair that that is comprised in the resolution, that these committee members be appointed by these various interests. As to the modus operandi, that has not been specified, but there are 10 days in which to do this, and in any way that

these various interests choose they may select the representatives who shall come at the time of the meeting on February 26 and appear as members of the committee.

Mr. HOOD. I should like the names of the committee members when they are appointed by the several interests. I should like that those names be sent to me at the Bureau of Mines, Pittsburgh, in order that I may communicate with each of you.

Mr. RICE. I think it would be very desirable if there would be some notification given by the different interests to the chairman in advance of the day of this meeting, so that he may know whether it is necessary to take up the question of the appointment of any person to a possible vacancy that might have to be filled.

Mr. DE WOLF. There has been, of course, some correspondence on the part of the Bureau of Mines officials in an effort to get a representative group at this meeting.

I would move that the chairman appoint now in a temporary capacity one representative from the men here present to represent each of these groups, and that the member so appointed be requested to obtain this list that has already been in use for his guidance, and that he take the responsibility for seeing that the men of his particular interest get organized.

The motion was carried.

Chairman FOHL. The committee will be announced later. Is there any further business of the same nature? If not, the chair would like to suggest that possibly a general invitation should be extended to all people interested to be present at these committee meetings. What would be the sentiment of the conference on that subject?

Mr. RICE. I should think there might be some objection from the standpoint of a working body. It seems to me that the representatives of each interest should try to gather all the information they can from their respective bodies; but if they all attend the meetings, might not some major matters be sidetracked by some of the minor features? I think it might be objectionable to have open meetings.

Mr. LELAND. I suppose when the committee has completed its labors it will report to a larger meeting, which will be open, and there will then be opportunity to discuss this, and that will be the time to invite every one to be present.

Chairman FOHL. Are there any further remarks?

Mr. RICE. If there is no further discussion, I would suggest that before these gentlemen go it would be desirable to take up the question of the appointment of this temporary committee. This is merely a suggestion to help the chairman.

Chairman FOHL. As the chair understands it, there are six groups of interests—the coal miners, the Bureau of Mines, the gas and oil

interests, the State inspectors, and the geologists. For the subcommittee the chair appoints the following: Mr. Schluederberg, Mr. Rice, Mr. Corrin, Mr. Crocker, Mr. Young, and Mr. DeWolf. Is there any further business before the conference?

Mr. RICE. Before the meeting closes, I wish to express on behalf of my colleagues of the Bureau of Mines their appreciation of the way in which this meeting has been conducted, and their appreciation of the response of the different interests in sending representatives to this meeting. I will add what I have several times repeated during the conference, that the bureau has no wish in any way to dominate or control the result. Its purpose was to start the ball moving and let it go of its own impetus. I think the thanks of the meeting are due to our chairman for his able handling of this meeting. I therefore propose a rising vote of thanks to the chairman and to the secretary of this meeting.

The motion, having been duly seconded and put to vote by Mr. Wilson, was carried unanimously.

Chairman FOHL. The chairman and secretary in turn will rise and acknowledge it. Is there any further business to come before the conference? If not, we are ready for a motion to adjourn.

Mr. Rice moves that we adjourn to meet at the possible future call of the chairman for the committee.

At 12.25 p. m. adjourned.

PUBLICATIONS ON PETROLEUM TECHNOLOGY.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.:

BULLETIN 19. Physical and chemical properties of the petroleums of the San Joaquin Valley, Cal., by I. C. Allen and W. A. Jacobs, with a chapter on analyses of natural gas from the southern California oil fields, by G. A. Burrell. 1911. 60 pp., 2 pls., 10 figs.

TECHNICAL PAPER 3. Specifications for the purchase of fuel oil for the Government, with directions for sampling oil and natural gas, by I. C. Allen. 1911. 13 pp.

TECHNICAL PAPER 10. Liquefied products from natural gas; their properties and uses, by I. C. Allen and G. A. Burrell. 1912. 23 pp.

TECHNICAL PAPER 25. Methods for the determination of water in petroleum and its products, by I. C. Allen and W. A. Jacobs. 1912. 13 pp., 2 figs.

TECHNICAL PAPER 26. Methods of determining the sulphur content of fuels, especially petroleum products, by I. C. Allen and I. W. Robertson. 1912. 13 pp., 1 fig.

TECHNICAL PAPER 32. The cementing process of excluding water from oil wells, as practiced in California, by Ralph Arnold and V. R. Garfias. 1913. 12 pp., 1 fig.

TECHNICAL PAPER 36. The preparation of specifications for petroleum products, by I. C. Allen. 1913. 12 pp.

IMPORTANCE OF OIL AND NATURAL-GAS INDUSTRIES.

In order to bring out the importance of the subject discussed at the conference and in this report, the following figures giving the value of the natural gas, petroleum, and coal produced in the United States in 1911 are presented:

Value of the natural gas, petroleum, and coal produced in 1911, by States.[a]

	Value of natural gas.	Value of crude petroleum.	Value of coal.
West Virginia	$28,451,907	$12,767,293	$53,670,515
Pennsylvania	18,010,796	10,894,074	321,537,250
Ohio	9,367,347	9,479,542	31,810,123
Oklahoma	6,731,770	26,451,767	6,291,494
Kansas	4,854,534	608,756	9,645,572
New York	1,418,767	1,248,950	
Indiana	1,192,418	1,228,835	15,326,808
Texas	1,014,945	6,554,552	3,273,288
Louisiana, Alabama	858,145	5,668,814	19,079,949
California	800,714	38,719,080	[b] 23,297
Illinois	687,726	19,734,339	59,519,478
Kentucky	407,689	328,614	13,617,217
Arkansas			3,396,849
Colorado	295,858	228,104	14,747,764
Wyoming		[c] 124,037	10,508,863
South Dakota	16,984		
Missouri	10,496	[d] 7,995	6,431,066
North Dakota	5,738		720,499
Michigan	1,330	[e] 7,995	2,791,461
Tennessee	300		7,209,734
Iowa	70		12,663,507
Utah		[f] 124,037	4,248,666
Washington			8,174,170
Virginia			6,254,804
Montana			5,342,168
Maryland			5,197,066
New Mexico			4,525,925
Georgia and North Carolina			246,448
Oregon			108,033
Idaho and Nevada			4,872
Total	74,127,534	[g] 134,044,752	626,366,876

[a] Mineral resources of the United States for 1911.
[b] Includes Alaska.
[c] Includes Utah.
[d] Includes Michigan.
[e] Includes Missouri.
[f] Includes Wyoming.
[g] Figures representing combined output of Utah and Wyoming and combined output of Missouri and Michigan counted only once.

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Bulletin 66

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

TESTS OF PERMISSIBLE EXPLOSIVES

BY

GLARENCE HALL
AND
SPENCER P. HOWELL

WASHINGTON
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1913

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TESTS OF PERMISSIBLE EXPLOSIVES.

By CLARENCE HALL and SPENCER P. HOWELL.

INTRODUCTION.

The tests and studies begun by the United States Geological Survey in the fall of 1908 with a view to lessening the accidents attending the use of explosives in coal mining are being continued by the Bureau of Mines at the Pittsburgh experiment station and in the field. One hundred and six applications for the testing of 209 different explosives had been received at the Pittsburgh station prior to March 1, 1913, and on that date 97 explosives that had passed the required tests were on the "permissible" list issued by the bureau. This bulletin gives the results of tests of all permissible explosives tested between May 15, 1909, and March 1, 1913. Bulletin 15 entitled "Investigations of Explosives used in Coal Mines" contains the results of tests of 17 permissible explosives tested prior to May 15, 1909. A copy of the test requirements and the names of explosives on the permissible list dated March 1, 1913, are appended to this bulletin.

An explosive is considered permissible for use in coal mines when it is similar in all respects to the sample that passed the tests required by the bureau, and when it is used in accordance with the conditions prescribed.

The physical tests reported in this bulletin were conducted by Messrs. H. F. Braddock, A. J. Strane, J. W. Koster, A. S. Crossfield, A. J. Hazlewood, A. B. Coates, and J. E. Tiffany at the Pittsburgh experiment station of the bureau.

The act of Congress making appropriations for sundry civil expenses of the Government for the fiscal year ending June 30, 1912, authorized the Secretary of the Interior to charge a fee covering the actual necessary expenses involved in making such tests by the Bureau of Mines as might be approved by him. After careful consideration had been given to the actual necessary expenses involved in the testing of explosives to determine their permissibility for use in coal mines, the sum of $150 was approved by the Secretary of the Interior as a reasonable charge for each complete official test.[a] During the first year this law was in effect the fees collected amounted to more than $6,000.

[a] Fees for testing explosives and conditions under which explosives will be received for test, Schedule 1, Bureau of Mines, 1913.

YEARLY SALES OF SHORT-FLAME EXPLOSIVES.

In the past few years there has been a rapid increase in the use of short-flame explosives in the coal mines of the United States. The following table shows the quantities of such explosives sold during each year from 1901 to 1912, inclusive:

Yearly sales of short-flame explosives in the United States, 1901–1912.

Year.	Quantity sold.	Year.	Quantity sold.
	Pounds.		*Pounds.*
1901	.....	1907	2,095,244
1902	11,300	1908	2,108,610
1903	288,661	1909	8,942,857
1904	608,270	1910	11,820,836
1905	1,031,300	1911	13,428,239
1906	1,533,575	1912	18,149,977

Unlike Government bureaus in some foreign countries, the Bureau of Mines has no authority to compel the use of certain explosives or to prescribe the conditions under which they must be used. Nevertheless, through the cooperation of State mine inspectors, manufacturers of explosives, coal operators, and miners the explosives that have been tested by the Bureau of Mines and designated permissible explosives are now used in every coal-mining State.

QUANTITY OF PERMISSIBLE EXPLOSIVES SOLD IN THE DIFFERENT COAL FIELDS.

The following table shows how rapidly the use of permissible explosives increased in the different coal fields in the United States during four years.

Quantity of permissible explosives sold in different coal fields in the United States, 1909–1912.

Coal fields and regions.	1909	1910	1911	1912
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*
1. Pennsylvania anthracite field	894,294	1,486,100	1,917,412	2,177,172
2. Northern Appalachian region [a]	5,976,625	5,967,216	6,350,272	9,267,149
3. Southern Appalachian region	1,434,950	3,188,785	3,377,268	3,920,125
4. Eastern interior field	6,955	165,975	337,012	733,940
5. Western interior field	254,475	196,560	255,050	439,825
6. Rocky Mountain region	146,700	808,200	1,177,075	1,488,789
7. Pacific coast region	228,858	8,000	14,150	122,977
	[b] 8,942,857	11,820,836	13,428,239	18,149,977

[a] Not including Pennsylvania anthracite field.
[b] Includes 344,830 pounds of short-flame explosives, the permissibility of which had not been determined.

The quantity of permissible explosives used in coal mining during the past few years no doubt replaced twice that weight of more dangerous explosives.

At many mines shot firers are employed when permissible explosives are used, and thus a more intelligent use of the explosives is

insured. Furthermore, a shot firer uses less explosive for bringing down coal than a miner who loads and fires his own shots.

Although the amount of permissible explosives used in this country is larger than in some foreign countries, it still represents only a small percentage of the total amount of explosives used in coal mining. Mines that are considered gaseous generally use permissible explosives. In some fields, if a certain part of a mine, such as the advance workings, is generating gas, or if inflammable coal dust is present in quantities considered dangerous, permissible explosives are used there, and black blasting powder or dynamite is used in all other parts of the mine.

The introduction of permissible explosives and their successful application to coal mining have had gratifying results. Few, if any, accidents can be directly attributed to the use of permissible explosives, in spite of the fact that no explosive can be considered entirely safe under all mining conditions. It has been shown that when permissible explosives are used in greater quantities than recommended by the bureau, or when they are not used in accordance with the other requirements, there is a possibility of a resultant fire-damp or coal-dust explosion. Moreover, if permissible explosives, when introduced at a coal mine, are placed in the hands of miners who are unfamiliar with their use, the results obtained are liable to be unsatisfactory until the miners have learned to use the explosives to best advantage.

FATALITIES IN COAL MINES.

The death rate in the coal mines of the United States had shown an increasing tendency up to the year 1908. In the spring of 1909 the bureau issued its first list of permissible explosives, and it is gratifying to note that the increase in the use of permissible explosives during the past four years is coincident with a decrease of fatal accidents in the coal mines. The following table shows the quantity of permissible explosives sold and the number of men killed in the coal mines of the United States from 1909 to 1912, inclusive:

Quantity of permissible explosives sold and number of men killed in coal mines, 1909–1912.

Year.	Quantity of permissible explosives sold.	Number of men killed.[a]	
		Per 1,000 employed.	Per 1,000,000 short tons mined.
	Pounds.		
1909	[b] 8,942,857	4.00	5.79
1910	11,820,836	3.92	5.66
1911	13,428,239	3.73	5.48
1912	18,149,977	3.15	4.29

[a] Horton, F. W., Coal-mine accidents in the United States, 1896-1912, with monthly statistics for 1912: Technical Paper 48, Bureau of Mines, 1912, p. 8.
[b] Includes 344,830 pounds of other short-flame explosives.

The greater interest in mine safety and the better enforcement of laws and regulations by State mine inspectors have been responsible to a large extent for the reduction in the death rate in the coal mines of this country, but it is believed that the use of permissible explosives has been the important factor in lessening the death rate from gas and coal-dust explosions.

The use of unsafe explosives in coal mining, as well as the improper use of such explosives, is responsible for many of the fatal accidents accredited to mine fires, falls of roof and coal, and other causes. It is known that when black blasting powder was used mine fires were a common occurrence in certain mines working a high-volatile coal, but when black blasting powder was replaced by permissible explosives mine fires resulting from shots were eliminated. Permissible explosives when properly used do not damage the roof such as is the case when black blasting powder is used; consequently, accidents due to falls of roof are lessened by the use of permissible explosives. The actual fall of rock or coal may not occur at the time of firing the charge, but heavy charges of black blasting powder weaken the walls and roof and start cracks that impair the supports of the roof, and considerable time may elapse after a blast before the roof falls. Another advantage in the use of permissible explosives is that the miner can set the props supporting the roof closer to the face of the workings than when black blasting powder is used, and thus insure better protection against roof falls.

APPARATUS AND METHODS FOR PHYSICAL TESTS OF EXPLOSIVES.

The apparatus used and the methods of testing explosives at the Pittsburgh experiment station of the Bureau of Mines have been described in Bulletin 15, Investigations of Explosives Used in Coal Mines. The new test requirements for permissible explosives as adopted by the bureau and certain new apparatus recently devised are described in detail in this bulletin.

The tests made at the Pittsburgh experiment station may be briefly summarized as follows:

PHYSICAL EXAMINATION.

This includes the determination of the average diameter, length, and weight of the cartridge, whether or not the cartridge has been redipped, the apparent specific gravity of the cartridge as determined by sand, and the color and consistency of the explosive.

BALLISTIC PENDULUM.

This apparatus consists of two essential parts, a cannon in which the charge is fired and a pendulum which receives the impact of the products of combustion and of the stemming. The pendulum is a

12.2-inch United States mortar, weighing 31,600 pounds, which is suspended by means of a stirrup from two cast-steel saddles fitting over a steel supporting beam that rests on nickel-steel knife edges on steel bearing-plates supported by concrete piers. The swing of the mortar when the cannon is discharged into it is measured by an automatic recording device and is taken as a measure of the propulsive effect of the explosive.

GAS AND DUST GALLERY.

This gallery consists of a steel cylinder 100 feet long and 6⅔ feet in diameter, into one end of which a cannon, such as is used in the ballistic pendulum tests, is discharged. The gallery may be filled with mixtures of gas and air, coal dust and air, or of gas, coal dust, and air, and the liability of the explosive fired in the cannon igniting such mixtures is determined by test.

RATE OF DETONATION APPARATUS.

The rate of detonation of an explosive is determined by inserting cartridges of the explosive in a tube of thin sheet iron that is suspended within a firing chamber. Into one end of the file of cartridge in the tube a detonator is inserted, and through points in the file a predetermined distance apart are led the wires of an electric circuit. By means of an extremely accurate recording device (Mettegang's recorder) the time required for the passage of the explosion wave through the distance between the two electric wires can be determined to within 1/4,300,000 second.

FLAME-TEST APPARATUS.

This apparatus is used to record by photography the relative lengths and durations of the flames produced by explosives, the test being based on the belief that the longer the flame and the longer the time it endures, the greater is the chance that such a flame, when shot into the air of a coal mine, will ignite inflammable mixtures of gas and air, coal dust and air, or gas, coal dust, and air. That the volume, density, conductivity, specific capacity, and temperature of the flame are, like the length and duration, important factors in determining its effect is recognized, but these must be ascertained and their effects measured by other means. Hence too great emphasis must not be placed on the value of the length and duration of the flame of explosives as an indication of their permissibility, for both the length and the duration of the flames of certain nonpermissible explosives are less than those of some of the permissible explosives, but the other variables, particularly the temperature of the flame, are of importance and must be taken into account. Furthermore, the particular manner in which the air offers resistance to the

flame may so laminate and disperse the flame that, in conjunction with other causes, its height and duration may be materially reduced. This air resistance accounts for certain variations in tests of the same explosive.

The flame-test apparatus consists essentially of a cannon, which is similar in all respects to those used in the ballistic pendulum and the gas and dust gallery tests, and a camera with suitable accessories for cutting off from a rapidly moving sensitive film all light rays except those emitted by the flame of the explosive. The length of flame is measured in milliseconds, or thousandths of a second, and the smallest time interval measurable by the apparatus is 0.0125 milliseconds.

EXPLOSION-BY-INFLUENCE TESTS.

The explosion-by-influence test determines the sensibility of an explosive to the explosion wave produced by the detonation of a mass of the same explosive. In this test two cartridges of the same explosive are placed a predetermined distance apart. One cartridge is fired by means of an electric detonator. If the second cartridge is detonated by the detonation of the first cartridge, the test is repeated with the second cartridge farther away. If the second cartridge is not detonated the test is repeated with the second cartridge nearer the first. By proceeding in this way, the minimum distance between cartridges at which the first cartridge does not detonate the second is established within 1 inch.

IMPACT MACHINE.

The impact machine is used to determine quantitatively the sensitiveness of explosives to explosion when they are struck by a known mass of steel of known velocity, while the explosive being tested rests on a steel surface. The essential parts of the apparatus are a case-hardened steel anvil that can be kept at a definite temperature by means of a water jacket, and a framework of steel rods, by means of which a steel hammer weighing 2,000 grams can be dropped from a precisely determined height on a charge of explosive on the anvil.

BICHEL PRESSURE GAGE.

The Bichel pressure gage is employed to determine the theoretical maximum pressure that an explosive would exert if exploded or detonated in a space that it fills completely, all the heat set free by the chemical reactions taking place being retained by the products of explosion. The apparatus also affords a means for collecting the resulting gaseous, liquid, and solid products. The essential features of the apparatus are two stout, steel cylinders in which the explosive is fired, and instruments by which the pressure within the cylinder is

recorded. From the pressure record the rate of dissipation of the heat from the explosion is ascertained.

CALORIMETER.

The calorimeter is designed to measure the quantity of heat set free by the explosion or detonation of a known quantity of explosive. It consists of a steel bomb in which the explosion takes place; a vessel for holding a known quantity of water in which the steel bomb is immersed; a wooden cylinder which supports the vessel and also insulates it; a stirring device by which the water in the immersion vessel is stirred so that the heat evolved by the explosion is uniformly distributed throughout the water; an extremely accurate thermometer for measuring the rise of temperature of the water following the explosion of the charge within the bomb, and scales for weighing the parts of the apparatus and the water used.

SMALL LEAD BLOCK TEST.

The small lead block test is intended to determine the compressive effect exerted by a detonating explosive when exploded unconfined on the upper end of a smaller cylinder of lead. In making the test, a small steel disk is placed on the cylinder to receive the immediate impact of the detonation. The deformation of the cylinder is taken as a measure of the disruptive effect of the explosive.

TRAUZL LEAD BLOCKS.

The Trauzl lead block test is designed to measure the comparative disruptive effect of explosives fired under moderate confinement. Equal weights of different explosives are confined in bore holes of definite dimensions made in lead blocks of prescribed character by means of a fixed quantity of stemming, and, when thus confined, are exploded by means of similar detonators. The measure of the test is the volume by which the cavity in the block is enlarged by pressure exerted by the explosive.

Apparatus other than the gallery used for testing the relative safety of coal-mining explosives and the methods of testing under accurately controlled conditions do not simulate actual conditions of practice, but it is generally recognized that such empirical tests have resulted in the use of safer explosives in coal mines.

PHYSICAL EXAMINATION.

In the physical examination of explosives the classification with respect to consistence has been elaborated in order to cover the characteristics of some of the recent explosives submitted for official

tests. All explosives are now classified with regard to the following characteristics:

- Consistence:
  - Granulation:
    - Structure:
      - Granular.
      - Fibrous.
      - Powdered.
      - Gelatinous.
    - Size:
      - Very fine.
      - Fine.
      - Coarse.
      - Very coarse.
  - Liquidness:
    - Very wet.
    - Wet.
    - Dry.
    - Very dry.
  - Compactness:
    - Very hard.
    - Hard.
    - Soft.
    - Very soft.
  - Cohesiveness:
    - Very cohesive.
    - Moderately cohesive.
    - Slightly cohesive.
    - Not cohesive.

BICHEL PRESSURE GAGES.

The construction of the pressure gages has not been altered since they were received at the experiment station, but the method of conducting tests has been slightly modified. Formerly, as in the tests described in Bulletin 15, the explosive was removed from its original wrapper and then wrapped in an envelope of tin foil before being placed in the gage. Gage tests of all explosives reported in this bulletin were made by firing charges of explosives in their original wrappers. It has been found that the paper wrapper of explosives, which represents a comparatively large percentage by weight of the combustible material, is an important factor in the character of the gaseous products evolved when the explosives are burned or detonated. The following are the analyses of the gases

produced by firing in the gage equal charges of the same explosive with and without paper wrappers:

Gaseous products evolved on detonation of an explosive in Bichel pressure gage.

[Analyst, A. L. Hyde.]

Gaseous products.	With paper wrapper.	Without paper wrapper.
	Per cent.	*Per cent.*
Carbon dioxide	27.3	51.4
Carbon monoxide	26.9	5.0
Oxygen	.0	.0
Hydrogen	18.0	2.2
Methane	.4	.3
Nitrogen	27.4	41.1

The analyses show that when the explosive was fired in its original wrapper there was an increase in the percentage of carbon monoxide formed. Other tests made with different kinds of explosives confirmed these results.

GASEOUS PRODUCTS EVOLVED ON DETONATION OF EXPLOSIVES IN FIELD TESTS.

The results of tests made in the field indicate that the gaseous products evolved on the detonation of all coal-mining explosives, together with the gases produced by the coal itself, are extremely poisonous and inflammable. The analyses of mine-air samples collected after shots in coal mines show that more carbon monoxide is formed when explosives are fired under actual working conditions than when fired in the gage. Some of the fine coal left in the drill hole is ignited and causes the increased amount of carbon monoxide formed.

It has been generally conceded that the permissible explosives that are deficient in oxygen and have passed all test requirements in metal cannons would produce a lower temperature when fired in a drill hole in coal, for the reason that any further addition of carbon would result in an increase of carbon monoxide, which would cause a reduction of the flame temperature. But the question has been raised whether the permissible explosives having an excess of oxygen would produce a higher temperature when fired in coal because of the possibility of the excess of oxygen uniting with just enough of the coal to form carbon dioxide. The results of the investigation of one explosive having an oxygen excess have not borne out this contention. On the other hand, all indications point to the fact that explosives of this kind are equally safe when fired in coal. Analyses of the gases produced from an explosive having an oxygen excess when fired in

the gage and also the gaseous products of the same explosive when fired under actual working conditions are given in the following table:

Gaseous products evolved on detonation of an explosive in Bichel pressure gage and in field tests.

[Analyst, A. L. Hyde.]

Gaseous products.	Gage test.		Field test.[a]
	With paper wrapper.	Without paper wrapper.	With paper wrapper.
	Per cent.	*Per cent.*	*Per cent.*
Carbon dioxide	30.0	24.3	22.6
Carbon monoxide	.0	.0	11.4
Oxygen	2.2	8.8	.0
Hydrogen	.0	.0	6.8
Methane	.7	.4	.7
Nitrogen	67.1	66.5	58.5

[a] Computed on an air-free basis; the methane produced from the coal was subtracted to make the methane percentage correspond to that in the gases from samples tested in the gage.

The analysis of the gases produced in field test shows a decrease in carbon dioxide and the production of a considerable amount of carbon monoxide. The field samples also show the presence of 6.8 per cent of hydrogen. On the assumption that this hydrogen was produced at the expense of some of the water in the products of combustion, the formation of less water and more hydrogen necessarily being accompanied by a smaller evolution of heat, its presence is further evidence of the reduction of flame temperature under actual working conditions. This reduction of temperature would offset any increase in the length of duration of the flame caused by the coal entering into the explosive reaction.

NEW REQUIREMENT WITH RESPECT TO THE KIND AND QUANTITY OF GASES EVOLVED ON DETONATION OF PERMISSIBLE EXPLOSIVES.

A list of permissible explosives issued by the bureau January 1, 1911, contained the names of a few explosives that on detonation in the Bichel gage developed large quantities of carbon monoxide. In tests of these explosives in coal mines it was found that when the usual charge was fired in rooms no appreciable amount of carbon monoxide was detected in samples of the mine air taken at the face immediately after the shot. However, when tests were made in narrow entries in coal mines, ahead of the air, all mine-air samples taken immediately after firing charges of 680 grams (1½ pounds) showed the presence of carbon monoxide, and one sample contained as much as 0.11 per cent of the gas.

Other permissible explosives that on detonation in the gage evolved less carbon monoxide were tested in coal mines. These explosives

did not appreciably vitiate the mine air in rooms or in narrow entries even when there was no circulation of air near the face. The highest percentage of carbon monoxide found in any of these mine-air samples taken immediately after the shot was 0.02.

It has been found, too, that all explosives which on detonation evolve 158 liters (5½ cubic feet) or more of permanent poisonous gases from a 680-gram (1½-pound) charge as determined by gage tests, when used in 680-gram (1½-pound) quantities in narrow entries in coal mines, ahead of the air, may for several minutes after a shot so vitiate the air near a face as to make it harmful. Of course, the conditions mentioned are unfavorable to any explosive and are abnormal in that there is no circulation of air near the face. However, as such conditions actually exist in some coal mines, all explosives that produce 5½ cubic feet (158 liters) or more of poisonous gases from a 1½-pound charge have been eliminated from the permissible list.

BALLISTIC PENDULUM.

Observations and tests in the field have shown that the ballistic pendulum and pressure gages measure the propulsive effect which approximately corresponds to the heaving or pushing force of explosives more nearly than any of the other apparatus used in testing explosives at the Pittsburgh experiment station. The tests with small lead block, the Trauzl lead block, and the rate-of-detonation apparatus measure approximately the disruptive (shattering) effect of explosives. It has been impossible to devise a single piece of apparatus for measuring both the propulsive and disruptive effects of explosives, and it is questionable if the combined effects determined by a single apparatus and expressed as the total energy of an explosive would be of any practical value. The useful work explosives can do depends on the material to be blasted and the method of application, etc. On account of varying conditions, certain characteristics of an explosive may be of more importance in some kinds of blasting than in others. Evidently if each of the fundamental characteristics of explosives was expressed in comparative figures the information would be a useful guide in selecting an explosive for a particular work. For this reason comparative figures are given for all tests of explosives reported in this bulletin.

NEW REQUIREMENTS WITH RESPECT TO THE EFFICIENCY OF PERMISSIBLE EXPLOSIVES.

During the past two years several permissible explosives have been withdrawn from the market by their manufacturers. All explosives that are no longer being manufactured, as well as those that have shown unfavorable results in field tests, have been eliminated from the list.

In the ballistic-pendulum test for determining the propulsive effect of an explosive the standard deflection of the pendulum is that produced by one-half pound (227 grams) of 40 per cent nitroglycerin dynamite. Some explosives do not produce an equal deflection unless they are used in charges of 1 pound (454 grams) or more, and it has been found that these explosives must be used in charges of more than 1½ pounds in order to satisfactorily break down coal. It is believed that in gaseous or dusty coal mines no explosive can be used safely in quantities greater than 1½ pounds per shot. Consequently all explosives that under normal conditions of mining can not be satisfactorily used in charges of 1½ pounds or less have been eliminated from the permissible list.

CALORIMETER.

In tests to determine the quantity of heat set free all explosives are now fired in their original wrappers. By determining the quantity of heat evolved in the calorimeter, the potential energy of explosives can be computed. The potential energy thus obtained represents the maximum work an explosive can theoretically accomplish, but as the useful work resulting from the use of explosives is a small percentage of the maximum, and varies considerably with the different classes of explosives, knowledge of the potential energy of explosives is of little value in selecting a suitable explosive.

Knowledge of the quantity of heat evolved on explosion is useful in determining the temperature of explosion, but too great an emphasis must not be placed on flame temperature alone as a measure of the permissibility of an explosive for use in coal mines. One explosive may develop a lower temperature on explosion than another, yet the duration of the flame may be such as to render the explosive of lower flame temperature unsuitable for use in coal mines.

A drill hole in coal or rock when filled with a charge of explosive is practically free of air; therefore these conditions must necessarily be simulated in experimental tests. In the calorimeter the air is almost entirely removed when explosives are tested. Explosives sometimes detonate incompletely and evolve oxides of nitrogen. It is generally recognized that a vacuum prevents the explosion of powder, and that high explosives are more likely to detonate incompletely when fired in a vacuum, especially when the charging density is low. It was therefore sometimes necessary to make several calorimeter tests in order to obtain satisfactory results. An incomplete detonation or the evolution of oxides of nitrogen in a calorimeter test should not be interpreted as an unfavorable result, for the reason that in practice explosives are never fired in a vacuum.

GAS AND DUST GALLERY NO. 1.

The position of the cannon in the gas and dust gallery No. 1 has been changed (see Pl. I, *A*). Formerly the cannon was embedded in the concrete head of the gallery and the replacement of worn-out cannons was difficult and expensive. That end of the gallery which was embedded in a concrete head is now closed by a steel head having in the center a hole 12 inches in diameter. The cannon is mounted on a 4-wheel truck which runs on a 30-inch (76.2-centimeter) gage track, and is placed so that the axis of its bore hole coincides with the longitudinal axis of the gallery. After the cannon is loaded it is rolled against the steel head and a close fit is made by means of a rubber gasket. A concrete arch surrounds the cannon to prevent pieces of metal from flying in the event of the cannon bursting. Behind the truck and 19 feet from the head of the gallery is a concrete barricade. The mouth of the cannon is in the same relative position as when the cannon was embedded in the concrete head. The change made does not affect the tests, while, on the other hand, a worn-out cannon can be easily replaced. It has been found that all explosives which pass official tests 1, 3, and 4 always pass tests 2 and 5. In view of this fact tests 2 and 5 have been discontinued as they were not considered discriminative.

CONSTRUCTION OF CANNON.

The principal expense involved in testing explosives is the cost of the cannon used. Various materials and methods of construction have been tried in an endeavor to obtain cannon which would best resist the shattering effects of the explosive. Black blasting powder and other slow-burning explosives have little effect on a cannon, but detonating explosives invariably disrupt it after a certain number of shots. Six thousand four hundred and eighty-six shots were fired in the first 46 cannons used at the Pittsburgh experiment station, or an average of 141 shots per cannon. Of these cannon 12 were relined with forged steel liners varying in diameter from 5 to 6 inches, so that the average number of shots per liner was 112. With the type of cannon now used it is possible to use two and sometimes three extra liners with each cannon, thus increasing the number of shots that can be fired in each cannon and materially reducing the expense. The first cannons used at the testing station were of the built-up type, the jacket being of cast low-carbon steel and the liner of forged nickel steel. Later solid steel forgings having the same length and diameter as the built-up cannons were tried. Different requirements covering the composition and physical properties of the materials were specified in the purchase of the solid forgings. These

forgings made with varying amounts of carbon and nickel were tried, and one cannon was made containing 0.20 per cent of vanadium. The solid steel forgings proved less resistant than the built-up cannons, and hence the use of such forgings was discontinued. A wire-wound cannon designed by the Ordnance Department of the United States Army and manufactured at Watervliet Arsenal burst after a few limit-charge tests. The jacket of the built-up cannons now used is a casting of soft steel thoroughly annealed. The liner is a soft low-carbon steel forging which elongates when explosives are fired within it. Hence the jacket is not subjected to the strain it would receive if elongation did not take place.

NONMETALLIC LINERS.

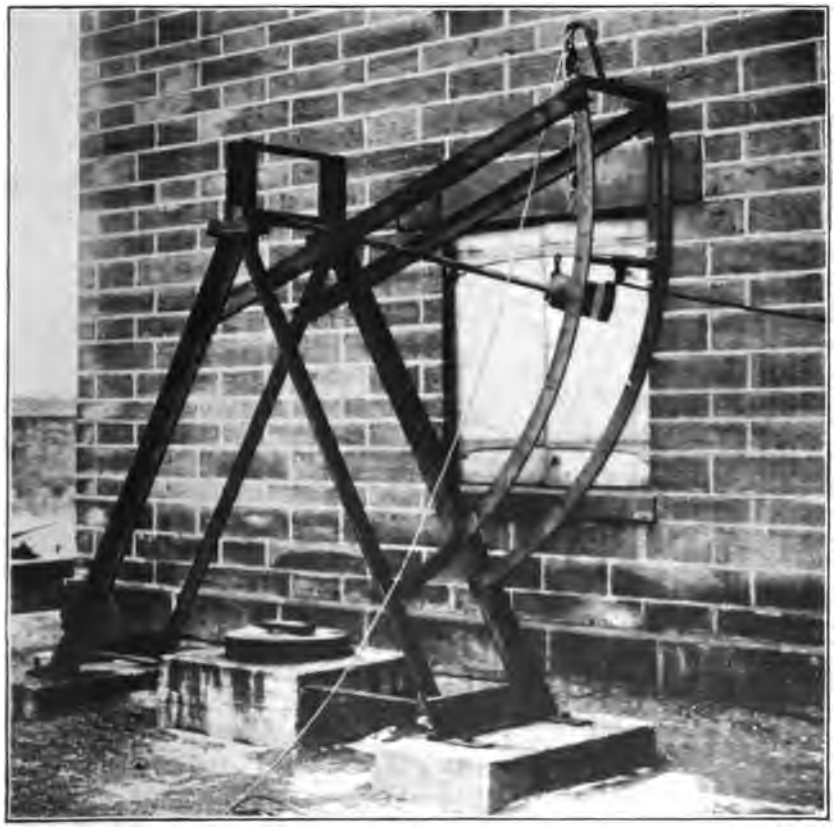
Several tests have been made with small nonmetallic liners in an endeavor to prevent the breaking of liners and jackets by explosions. Portland cement, plaster of Paris, and cement liners containing crushed coal were tried, but it was found that, although the life of a cannon might possibly be prolonged, the liners were pulverized, and the pulverized material projected into the gallery had a tendency to reduce the temperature of the gaseous products of explosion. Consequently, the use of such liners was not deemed desirable in determining the permissibility of explosives. In several instances nonpermissible explosives passed one or more of the gas and dust gallery tests when nonmetallic liners were used, but failed under the regular conditions of testing.

ADOBE SHOTS IN COAL MINES.

Results of tests made in the gallery indicate that no explosive is safe for "adobe shots"[a] in breaking up large pieces of rock or coal or in breaking down stoppings in coal mines where either gas or dry inflammable dust is present in quantities or under conditions that indicate danger. Black blasting powder, even when covered with large quantities of clay, can not be safely or effectively used for this work. Ordinary dynamites are equally unsafe, even when put in a bucket of water placed on the material to be broken they give sufficient flame to cause the explosion of an inflammable mixture of gas and air. Permissible explosives have been found to be safer than black blasting powder or ordinary dynamite for use in "adobe shots," but the charges used must be small and must be well covered with moist clay. However, all permissible explosives, when fired under these conditions in the presence of a mixture of gas and air containing 8 per cent of gas (methane) are likely to cause ignition and consequently "adobe shots" should never be fired in the pres-

[a] For an "adobe shot" the explosive is placed on the thing to be broken. This method of using an explosive is also called "bulldozing."

A. GAS AND DUST GALLERY NO. 1.



B. PENDULUM FRICTION DEVICE.

ence of fire damp. In emergencies, such as in fighting mine fires or in recovery work when "adobe shots" are absolutely necessary, the working place should be carefully tested for gas before firing. The charge used should not exceed 1½ pounds and it should be well covered with moist clay.

MAXIMUM CHARGE PERMISSIBLE FOR USE IN COAL MINES.

The underlying reasons why one explosive passes and another fails when tested in the presence of inflammable gas and dust have been investigated at the Pittsburgh experiment station.

The results of tests indicate that any explosive fired in large quantities will cause ignition of inflammable gas and coal dust mixtures. An arbitrary charge, namely, 1½ pounds, has been established as the maximum amount of explosive to be used in making tests, and all explosives in order to be placed on the permissible list must pass the gas and dust test with this charge. A charge of 1½ pounds of explosive per drill hole should never be exceeded in practice; in good mining practice the charge need not exceed 1 pound, an amount which gives a greater factor of safety.

Explosives of many different compositions are now on the permissible list, but every one is designed to give on detonation a relatively low flame temperature and a flame of short duration. To ignite inflammable gas and coal dust mixtures a certain temperature, acting through a certain length of time, is required. The flame temperature of all explosives exceeds the ignition temperature of inflammable gas and dust mixtures, but fortunately the flame of a permissible explosive properly detonated is of such short duration that under the conditions of blasting down coal it does not ignite the inflammable mixtures. Manifestly, any factor that increases the duration of the flame temperature of a permissible explosive increases the danger in the use of that explosive. Such danger is present when an improper detonator is used or when any explosive is not used in accordance with the provisions prescribed by the Bureau of Mines.

NEW TESTING APPARATUS.

PENDULUM FRICTION MACHINE.

A new test, the sensitiveness of explosives to frictional impact (a glancing blow) has been adopted by the Bureau of Mines. The apparatus used (see Pl. I, *B*, and fig. 1) comprises a steel anvil and a swinging steel shoe. The anvil has a smooth face, 3¼ inches wide by 12 inches long, in the middle of which are three grooves for holding the charge of explosive. The radius of swing of the shoe is 6 feet 6¾ inches and the radius of curvature of its face is 10½ inches. The shoe can be faced with hardwood fiber or other material, and may be dropped from heights of 19.7 to 78.7 inches (½ to 2 meters). Weights of 2.2

to 44 pounds (1 to 20 kg.) are used in the tests. Details of the machine are shown in figure 1.

The test was devised to represent what may happen in mining when a cartridge of explosive becomes lodged in a drill hole and

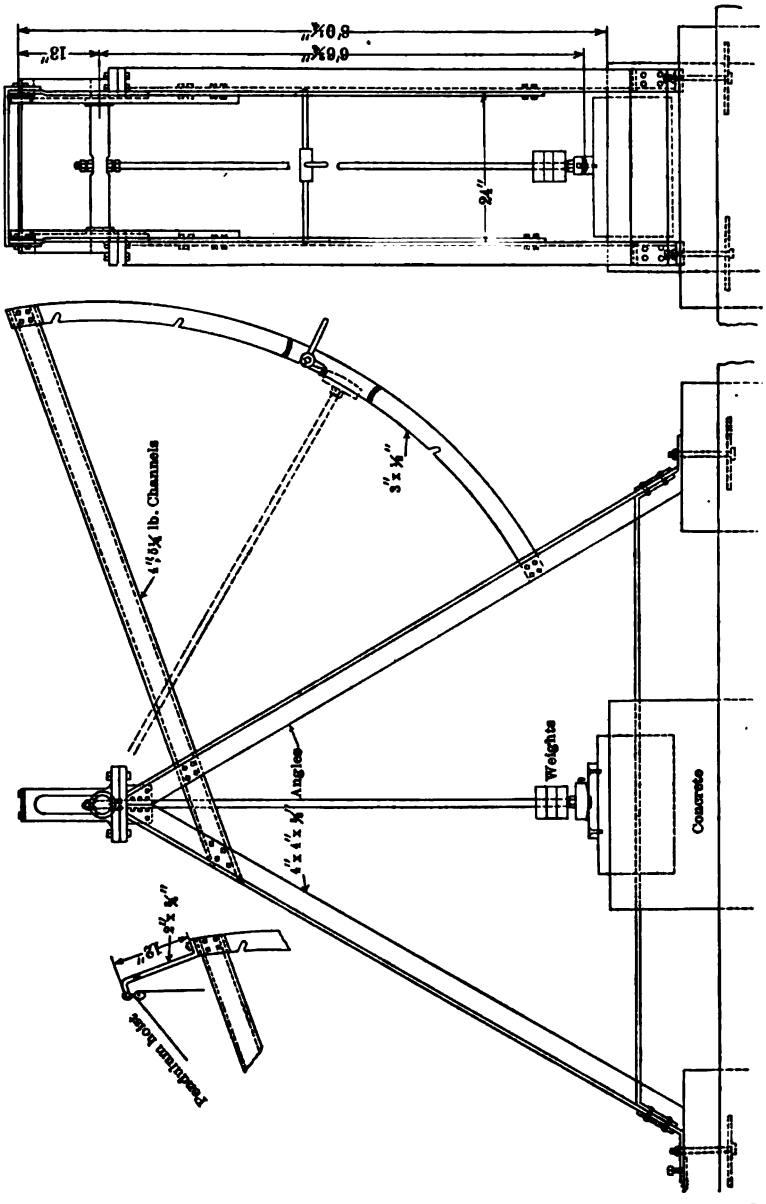


FIGURE 1.—Elevation and details of pendulum friction machine.

must be forced home by ramming. Some explosives have been found to be extremely sensitive to the frictional-impact test even when the shoe is faced with wood fiber. Consequently great care should be exercised with all explosives in charging drill holes.

LARGE IMPACT MACHINE.

A new and much larger impact machine (fig. 2) constructed somewhat on the same principle as the small impact machine already mentioned was recently installed at the Pittsburgh testing station. Like the small machine it is designed to determine the sensitiveness of explosives to explosion by direct impact. The falling weight used in the test is 200 kilograms (441 pounds); the maximum height from which the hammer can be dropped is 7.5 meters (24.6 feet); the charge of explosive used, 20 grams (309 grains), is placed on the anvil in the form of a flat disk 10 centimeters (3.94 inches) in diameter. The face of the anvil and the face of the plunger are 12.7 centimeters (5 inches) in diameter. Tests are conducted in the same way as with the small impact machine. The advantage of the large machine is that it permits the use of larger charges of explosives, thus minimizing the variables, such as the physical constitution of the explosive, that affect the results obtained with the small machine; the charge being larger more nearly represents the explosive, and the irregularities with respect to the composition and size of the ingredients of an explosive are minimized. Another advantage is that in the tests there is usually a sharp report or no explosion, whereas with an impact machine using a small quantity of explosive it is difficult sometimes to determine whether an explosive detonates completely or partly, or merely ignites. The greatest advantage of the large machine is that tests can be made with explosives when confined between two disks, 9.8 cm. in diameter, fitted into a piece of Shelby steel tubing the height of which is equal to the total thickness of the two disks. From comparative tests with large and small machines it is believed that the results obtained with the large impact machine when the explosives are confined more nearly represent the relative sensitiveness of explosives to direct impact.

CLASSES OF PERMISSIBLE EXPLOSIVES.

In order that the users of explosives may know the nature and characteristic component of each of the permissible explosives, and as an aid in the selection of an explosive to meet a specific requirement, the permissible explosives have been arranged in four classes. In classifying the permissible explosives, both the composition of the explosive and the behavior of the characteristic component material when stored or used were considered.

The four classes and the general characteristics of the permissible explosives are as follows:

Class 1, ammonium nitrate explosives.—To class 1 belong all the explosives in which the characteristic material is ammonium nitrate.

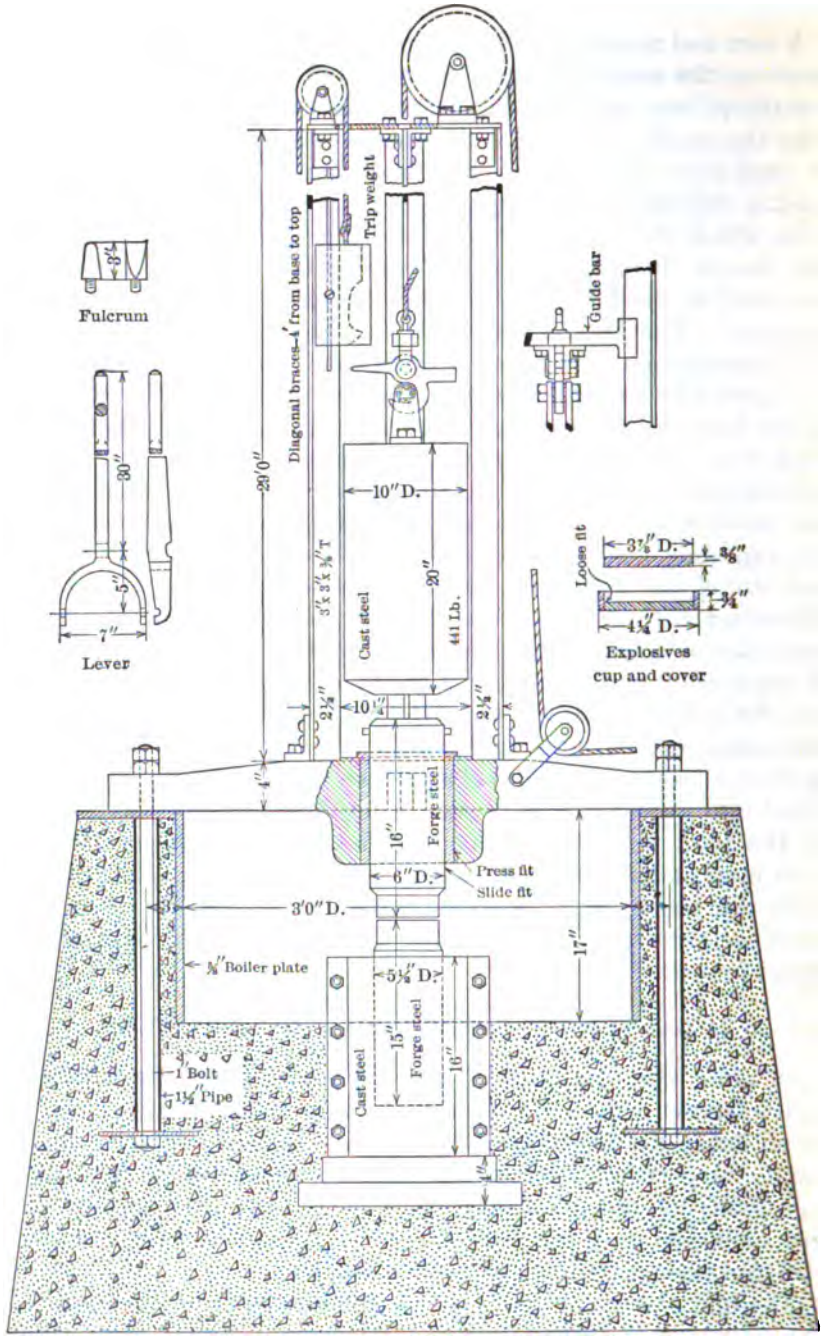


FIGURE 2.—Elevation and details of large impact machine.

The class is divided into two subclasses. Subclass *a* includes every ammonium nitrate explosive that contains a sensitizer that is itself an explosive. Subclass *b* includes every ammonium nitrate explosive that contains a sensitizer that is not in itself an explosive. The ammonium nitrate explosives of subclass *a* consist principally of ammonium nitrate with small percentages of nitroglycerin, nitrocellulose, or nitrosubstitution compounds which are used as sensitizers. The ammonium nitrate explosives of subclass *b* consist principally of ammonium nitrate with small percentages of resinous matter or other nonexplosive substances used as sensitizers.

All of the ammonium nitrate explosives readily absorb moisture from the atmosphere, and great care should be taken in storing them or in using them in damp places. They are not suitable for use in wet mines. If in such a mine a cartridge of an ammonium nitrate explosive is opened and its contents exposed for only a few hours to the damp atmosphere the explosive may deteriorate and later fail to detonate completely. The ammonium nitrate explosives when stored in well-ventilated magazines for only a few months have shown signs of deterioration, and nearly all explosives of this class after six months' storage at the Pittsburgh experiment station have either failed to detonate or have detonated incompletely. For this reason ammonium nitrate explosives should be obtained in a fresh condition and should be used as soon as possible after their receipt. When fresh, these explosives, if properly detonated, have the advantage of producing only small quantities of poisonous and inflammable gases, and are adapted for mines that are not unusually wet, and also for mines and working places that are not well ventilated.

Class 2, hydrated explosives.—To class 2 belong all explosives in which salts containing water of crystallization are the characteristic materials. The explosives of this class are somewhat similar in composition to the ordinary low-grade dynamites, except that one or more salts containing water of crystallization are added to reduce the flame temperature. They are easily detonated, produce only small quantities of poisonous gases, and most of them can be used successfully in damp working places.

Class 3, organic nitrate explosives.—To class 3 belong all the explosives in which the characteristic material is an organic nitrate other than nitroglycerin. The permissible explosives listed under class 3 are nitrostarch explosives. They produce small quantities of poisonous gases on detonation.

Class 4, nitroglycerin explosives.—To class 4 belong all the explosives in which the characteristic material is nitroglycerin. These explosives contain free water or an excess of carbon, which is added to reduce the flame temperature. A few explosives of this class contain salts that reduce the strength and shattering effect of the explosives

on detonation. Nitroglycerin explosives have the advantages of detonating easily and of not being readily affected by moisture. On detonation some of them produce as large quantities of poisonous and inflammable gases as black blasting powder, and for this reason they should not be used in mines or working places that are not well ventilated.

RATE OF DETONATION OF PERMISSIBLE EXPLOSIVES.

The energy developed by the detonation of permissible explosives, like that of other high explosives, depends on the change of the small solid and liquid particles of the explosive into large volumes of highly heated gases and on the rate of detonation or the rapidity with which these gases are formed. The force exerted by these gases is the means of producing useful effects. The rate of detonation is the governing factor in judging the efficiency of an explosive and it offers the best single means for selecting explosives suitable to meet the varying conditions of coal mining.

During the conversion of an explosive into solid, liquid, and gaseous products the cooling effect of the walls of the drill hole tends to lower the temperature of the gases so that the maximum theoretical temperature, or pressure, is never reached. The more nearly instantaneous the explosive reaction, all other conditions being equal, the greater the volume of highly heated gases produced, and the more violent the effect.

To meet the varying conditions of coal mining in this country the explosives manufacturers have devised explosives with rates of detonation that range from 4,750 to 14,560 feet (1,447 to 4,439 meters) per second.

SUGGESTIONS USEFUL IN SELECTING EXPLOSIVES.

It is hoped that with few exceptions the classification given will serve as a useful guide for comparing the practical value of permissible explosives. However, in order to select the most suitable, users of these explosives should conduct a series of experiments in the mine. The information given herein should eliminate the necessity of testing many different classes and brands of explosives.

It is evident that for certain work in which a shattering effect is desired, as in driving through or "brushing" rock, or in producing coal for coking purposes, the explosive reaction should be rapid. Hence, permissible explosives having a high rate of detonation should be selected. Similarly, for use in soft friable coal to produce lump or steam coal, selection should be made of a permissible explosive that detonates slowly and hence gives a more prolonged pressure. In medium hard coal an explosive having an intermediate rate of detonation would be expected to be most suitable, but is not always so. Coals vary in hardness and coal beds vary in the number

and position of the joints, partings, shale bands, etc. These facts have to be considered in mining.

An explosive having a very low rate of detonation is not always best suited for mining soft friable coal, because some of its energy may be lost by its gases escaping through cracks and fractures in the bed. Under such conditions an explosive having an intermediate rate produces the most economical results.

Another factor to be considered in connection with an explosive having a high rate of detonation or the use of a large charge of any explosive is the possible effect on the roof, or the strata overlying the coal bed. Large charges of explosives having a very high rate of detonation cause small fissures that may later necessitate extra timbering to prevent falls in rooms or entries, and thus make the operation of a mine more costly.

It is well known that the pressure developed by the detonation of explosives in a closed space is directly proportional to the charging density; in other words, a 1¾-inch drill hole loaded with 1¼-inch cartridges will produce on the walls of the drill hole about one-half as much pressure per square inch as it would if loaded with cartridges of 1¾-inch diameter. A limited experience indicates that explosives having a rapid rate of detonation will yield a larger proportion of lump coal if used in a hole of larger diameter than the cartridge. Such air spacing to reduce the shattering effect of an explosive is recommended by the Bureau of Mines, provided the charge is confined with moist clay stemming tamped to the mouth of the drill hole.

Other less desirable means of reducing the shattering effect of an explosive are the use of an improper detonator, reducing the amount of stemming, using an explosive that is frozen or partly frozen, using an explosive in cartridges of less diameter than those originally tested, and introducing foreign substances between the cartridges of an explosive; but these methods are all dangerous. They not only eliminate the safety qualities of the explosive but increase the chance of a resultant dust or gas explosion, and should not be adopted.

RESULTS OF TESTS WITH DETONATORS AND ELECTRIC DETONATORS.

Permissible explosives are detonated by means of detonators or electric detonators, which are graded by number according to the weight of fulminating charge. Different types of explosives require detonators of different strength. Detonators are usually fired with fuse. A detonator fitted with a means of firing it with an electric current is called an electric detonator. As an electric detonator is embedded in the explosives with which it is used and is isolated by the stemming, it is the safest means of igniting a charge of explosive in a gaseous mine.

One of the conditions prescribed by the Bureau of Mines for a permissible explosive is that it shall be fired by a detonator, preferably an electric detonator having a charge equivalent to that of the standard detonator used at the Pittsburgh experiment station. It is further required that this charge shall consist by weight of 90 parts of mercury fulminate and 10 parts of potassium chlorate (or their equivalents).

An investigation undertaken by the Bureau of Mines shows that the average percentage of failures of explosives to detonate is increased over 20 per cent when the lower grades of electric detonators were used instead of No. 6 electric detonators, and over 50 per cent when compared with No. 8 electric detonators.[a] However, when sensitive explosives were tested under the conditions most favorable to detonation, the same energy was developed irrespective of the detonator used. When tests were made with insensitive explosives under conditions which similate their use in blasting, the energy increased with the grade of the detonator used. For example, the average explosive efficiency of four different explosives was increased 10.4 per cent by using a No. 6 electric detonator instead of a No. 4 electric detonator, and 14.9 per cent by using a No. 8 electric detonator. The tests emphasize the importance of using explosives in a fresh condition, and, since this is not always possible, the importance of strong detonators in blasting to offset any deterioration of the explosive through ageing.

The results substantiate these conclusions: (1) That the explosive efficiency of the detonators made by any one manufacturer increases with the grade; (2) that No. 6 electric detonators of four different makes have practically the same explosive efficiency; and (3) each is considered equivalent to the Pittsburgh experiment station standard No. 6 electric detonator for use with permissible explosives in coal mines when the No. 6 grade is prescribed.

RESULTS OF TESTS OF PERMISSIBLE EXPLOSIVES.

ÆTNA COAL POWDER AA.

Explosive, Ætna coal powder AA.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Ætna Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 261 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, drab.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

[a] Hall, Clarence, and Howell, S. P., Investigations of detonators and electric detonators: Bull. 59, Bureau of Mines, 1913, p. 73.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 28, 1909.
Unit swing on this date, 2.65 inches.
Weight of charge, in grams, 230, 230, 230.
Swing, in inches, 2.50, 2.63, 2.46.
Average swing, in inches, 2.53.

2.53 : 2.65 : : 230 : (241).

Therefore the unit deflective charge of Ætna coal powder AA is 241 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 29	241	7.81	No ignition.	Sept. 30	241	.....	No ignition.
Do	241	8.11	Do.	Do	241	.....	Do.
Do	241	7.81	Do.	Do	241	.....	Do.
Do	241	7.81	Do.	Do	241	.....	Do.
Sept. 30	241	8.05	Do.	Do	241	.....	Do.
Do	241	8.05	Do.	Do	241	.....	Do.
Do	241	8.05	Do.	Do	241	.....	Do.
Do	241	7.87	Do.	TEST 4.			
Do	241	7.79	Do.				
Do	241	7.87	Do.	Sept. 27	[a] 680	4.44	No ignition.
TEST 3.				Do	[a] 680	4.19	Do.
				Do	[a] 680	4.20	Do.
Sept. 30	241	.....	No ignition.	Do	[a] 680	4.19	Do.
Do	241	.....	Do.	Do	[a] 680	4.19	Do.
Do	241	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 28	18.09	43.0	2,377
Sept. 29	17.55	43.0	2,450
Do	18.05	43.0	2,382

Average rate of detonation, 2,403 meters (7,880 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 11	13.50	17.05	6.50	0.325
Do	13.00	16.42	7.50	.375
Do	12.00	15.16	5.00	.250

Average height of flame, 16.21 inches.
Average duration of flame, 0.317 millisecond.

IMPACT TEST.

Date, January 5, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	10	1	No explosion.
18	1	Do.	11	3	Do.
16	1	Do.	11	1	Explosion.
12	1	Do.	10	4	No explosion.

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 166 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 14	6	Did not explode.	Dec. 14	4	Exploded.
Do	4	Do.	Dec. 16	5	Did not explode.
Do	2	Exploded.	Do	5	Do.
Do	3	Do.	Do	5	Do.
Do	4	Did not explode.			

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912-13).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Dec. 30	212	1.02	30.00	93.75	A	
Jan. 2	212	1.02	30.50	95.31	A	93.75
Do	212	1.02	29.50	92.19	A	
Dec. 28	212	1.02	29.00	90.62	B	
Dec. 30	212	1.02	30.25	94.53	B	93.23
Do	212	1.02	30.25	94.53	B	
Dec. 27	212	1.02	27.75	86.72	C	
Do	212	1.02	28.00	87.50	C	88.02
Dec. 28	212	1.02	28.75	89.84	C	

[a] Including 12 grams original wrapper.

P=1.911A+0.5B−1.411C=101.58 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.02. W=212 grams.

$M=\frac{VPS}{W}$=7,331 kilograms per square centimeter (104,270 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 12 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 10, 1912.

	Grams.
Solid	27.7
Liquid (water)	61.5
Gaseous	111.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C*	*Calories.*
Jan. 9	81.95	1.111	971.9
Do	81.95	1.106	967.5
Do	81.95	1.059	926.0

[a] Including 6 grams of original wrapper.

Average large calories per kilogram of explosive, 955.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters*
Mar. 10	63.50	56.00	7.50
Do	63.00	55.50	7.50
Do	63.50	55.25	8.25

Average compression, 7.8 millimeters (0.31 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	229	166	15
Do	63	236	173	15
Do	63	230	167	15

Average expansion of bore hole, 169 cubic centimeters (10.31 cubic inches).

ÆTNA COAL POWDER C.

Explosive, Ætna coal powder C.
Class 4, nitroglycerin.
Manufactured by the Ætna Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 316 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.22.
Color of explosive, drab.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 28, 1909.
Unit swing on this date, 2.65 inches.
Weight of charge, in grams, 340, 340, 340.
Swing, in inches, 2.64, 2.63, 2.59.
Average swing, in inches, 2.62.

2.62 : 2.65 : : 340 : (344).

Therefore the unit deflective charge of Ætna coal powder C is 344 grams.

GAS AND DUST GALLERY No. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Sept. 29	344	8.55	No ignition.	Oct. 1	344	.....	No ignition.
Do	344	8.55	Do.	Do	344	.....	Do.
Do	344	8.38	Do.	Do	344	.....	Do.
Do	344	8.38	Do.	Do	344	.....	Do.
Do	344	8.05	Do.	Do	344	.....	Do.
Do	344	8.05	Do.	Do	344	.....	Do.
Do	344	7.88	Do.	Do	344	.....	Do.
Do	344	7.96	Do.	TEST 4.			
Do	344	7.78	Do.				
Do	344	8.05	Do.	Sept. 27	[a] 680	4.44	No ignition.
TEST 3.				Do	[a] 680	4.39	Do.
				Do	[a] 680	4.52	Do.
Sept. 30	344	.....	No ignition.	Do	[a] 680	4.46	Do.
Oct. 1	344	.....	Do.	Do	[a] 680	4.35	Do.
Do	344	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 28	20.72	43.0	2,075
Sept. 30	19.36	43.0	2,221
Oct. 1	20.30	43.0	2,118

Average rate of detonation, 2,138 meters (7,010 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 29	17.50	22.11	9.00	0.450
Do	15.75	19.89	7.50	.375
Do	15.25	19.26	7.50	.375

Average height of flame, 20.42 inches.
Average duration of flame, 0.400 millisecond.

IMPACT TEST.

Date, May 6, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
12	1	Explosion.	8	1	Explosion.
10	1	Do.	7	2	No explosion.
8	1	No explosion.	7	1	Explosion.
9	2	Do.	6	5	No explosion.
9	1	Explosion.			

The maximum height at which no explosion occurs established at 6 centimeters (2.36 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 172 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 18	6	Did not explode.	Dec. 18	5	Did not explode.
Do	4	Do.	Do	5	Exploded.
Do	3	Exploded.	Do	6	Did not explode.
Do	4	Do.	Do	6	Do.
Do	5	Did not explode.			

The minimum distance at which no explosion occurs established at 6 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912-13).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 2	215	1.08	16.25	50.78	A	51.56
Do	215	1.08	16.25	50.78	A	
Jan. 3	215	1.08	17.00	53.12	A	
Dec. 26	215	1.08	15.50	48.44	B	48.18
Do	215	1.08	15.25	47.66	B	
Do	215	1.08	15.50	48.44	B	
Dec. 27	215	1.08	15.25	47.66	C	47.14
Do	215	1.08	15.00	46.88	C	
Do	215	1.08	15.00	46.88	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=56.11$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.08$. $W=215$ grams.

$M=\frac{VPS}{W}=4{,}228$ kilograms per square centimeter (60,140 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 11, 1912.	Grams.
Solid	81.7
Liquid (water)	11.8
Gaseous	102.6

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 10	63.50	52.50	11.00
Do	63.50	53.00	10.50
Do	63.50	53.00	10.50

Average compression, 10.7 millimeters (0.42 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	207	144	15
Do	63	208	145	15
Do	63	207	144	15

Average expansion of bore hole, 144 cubic centimeters (8.78 cubic inches).

BENTAL COAL POWDER NO. 1-A.

Explosive, Bental coal powder No. 1-A.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 7⅞ inches.
Average weight, 179 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.09.
Color of explosive, gray.
Consistency, powdered, very fine, very dry, very soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, December 4, 1911.
Unit swing on this date, 3.40 inches.
Weight of charge, in grams, 280, 280, 280.
Swing, in inches, 3.41, 3.40, 3.44.
Average swing, in inches, 3.42.

$$3.42 : 3.40 : : 280 : (278).$$

Therefore the unit deflective charge of Bental coal powder No. 1-A is 278 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Dec. 6	278	7.83	No ignition.	Dec. 14	278	.....	No ignition.
Do	278	7.91	Do.	Do	278	.....	Do.
Do	278	8.21	Do.	Do	278	.....	Do.
Do	278	7.91	Do.	Do	278	.....	Do.
Do	278	8.22	Do.	Do	278	.....	Do.
Do	278	8.31	Do.	Do	278	.....	Do.
Do	278	8.15	Do.	Do	278	.....	Do.
Do	278	8.06	Do.	TEST 4.			
Do	278	8.24	Do.				
Do	278	7.92	Do.	Dec. 12	[a]680	4.37	No ignition.
TEST 3.				Do	[a]680	4.42	Do.
				Do	[a]680	4.17	Do.
Dec. 14	278	.....	No ignition.	Do	[a]680	4.29	Do.
Do	278	.....	Do.	Do	[a]680	4.11	Do.
Do	278	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator No. 7.

Date (1911–12).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 30	18.60	44.0	2,366
Jan. 3	19.00	45.0	2,368
Do	19.60	45.0	2,296

Average rate of detonation, 2,343 meters (7,690 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 6	13.50	17.05	6.00	0.300
Do	10.00	12.63	5.25	.262
Do	9.00	11.37	3.75	.188

Average height of flame, 13.68 inches.
Average duration of flame, 0.250 millisecond.

IMPACT TEST.

Date, February 26, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
25	1	Explosion.	17	1	No explosion.
20	1	Do.	17	1	Explosion.
19	1	Do.	16	1	Do.
18	1	No explosion.	15	5	No explosion.
18	1	Explosion.			

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 177 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 1	3	Did not explode.	Apr. 1	4	Did not explode.
Do	2	Exploded.	Do	4	Do.
Do	3	Do.	Apr. 2	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912.)	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 29	214	1.09	28.00	87.50	A	
Apr. 1	214	1.09	28.00	87.50	A	87.24
Do	214	1.09	27.75	86.72	A	
Apr. 2	214	1.09	24.75	77.34	B	
Do	214	1.09	24.00	75.00	B	75.52
Apr. 3	214	1.09	23.75	74.22	B	
Apr. 4	214	1.09	23.75	74.22	C	
Do	214	1.09	24.00	75.00	C	74.74
Do	214	1.09	24.00	75.00	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=99.02$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.09$. $W=214.0$ grams.

$M=\frac{VPS}{W}=7{,}565$ kilograms per square centimeter (107,610 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 14 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, December 7, 1911.	Grams.
Solid	67.8
Liquid (water)	54.0
Gaseous	88.1

Compression of Small Lead Blocks.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters	*Millimeters.*	*Millimeters.*
Mar. 29	63.00	50.75	12.25
Do	63.50	51.50	12.00
Do	63.25	51.50	11.75

Average compression, 12.0 millimeters (0.47 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	°C.
Feb. 26	62	188	126	15
Do	62	190	128	15
Do	62	188	126	15

Average expansion of bore hole, 127 cubic centimeters (7.75 cubic inches).

BENTAL COAL POWDER NO. 2.

Explosive, Bental coal powder No. 2.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 174 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, gray.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, September 12, 1910.
Unit swing on this date, 3.20 inches.
Weight of charge, in grams, 270, 270, 270.
Swing, in inches, 3.13, 3.07, 3.19.
Average swing, in inches, 3.13.

3.13 : 3.20 : : 270 : (276).

Therefore the unit deflective charge of Bental coal powder No. 2 is 276 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.			
	Grams.	*Per cent.*	
Sept. 13	276	8.05	No ignition.
Do	276	7.90	Do.
Do	276	8.15	Do.
Do	276	8.09	Do.
Do	276	8.20	Do.
Do	276	8.17	Do.
Do	276	8.09	Do.
Do	276	8.17	Do.
Do	276	8.06	Do.
Do	276	7.87	Do.
TEST 3.			
Sept. 13	276	.....	No ignition.
Do	276	.....	Do.
Do	276	.....	Do.
TEST 3—Con.			
Sept. 13	276	.....	No ignition.
Do	276	.....	Do.
Do	276	.....	Do.
Do	276	.....	Do.
Do	276	.....	Do.
Do	276	.....	Do.
Do	276	.....	Do.
TEST 4.			
Sept. 12	[a] 680	4.16	No ignition.
Do	[a] 680	4.08	Do.
Do	[a] 680	4.03	Do.
Do	[a] 680	4.03	Do.
Do	[a] 680	4.13	Do.

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 12	30.52	43.5	1,425
Do	30.15	43.5	1,443
Do	29.55	43.5	1,472

Average rate of detonation, 1,447 meters (4,750 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 20	10.50	13.26	3.50	0.175
Do	12.25	15.47	5.50	.275
Do	8.50	10.74	3.50	.175

Average height of flame, 13.16 inches.
Average duration of flame, 0.208 millisecond.

Impact Test.

Date, February 4, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
24	1	Explosion.	17	1	Explosion.
20	1	Do.	16	5	No explosion.
18	1	Do.			

The maximum height at which no explosion occurs established at 16 centimeters (6.3 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 166 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 13	6	Did not explode.	Feb. 13	0	Exploded.
Do	4	Do.	Do	1	Do.
Do	2	Do.	Do	2	Did not explode.
Do	1	Do.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 8	214	1.03	27.25	85.16	A	84.64
Do	214	1.03	27.00	84.38	A	
Do	214	1.03	27.00	84.38	A	
Feb. 10	214	1.03	25.25	78.91	B	79.17
Do	214	1.03	25.00	78.12	B	
Do	214	1.03	25.75	80.47	B	
Do	214	1.03	24.25	75.78	C	75.52
Feb. 11	214	1.03	24.50	76.56	C	
Do	214	1.03	23.75	74.22	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=94.77$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.03$. $W=214$ grams.

$M=\frac{VPS}{W}=6{,}842$ kilograms per square centimeter (97,320 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 16 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 27, 1911.	Grams.
Solid	42.7
Liquid (water)	67.0
Gaseous	94.4

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 25	63.50	57.00	6.50
Do	63.50	57.25	6.25
Do	63.50	57.00	6.50

Average compression, 6.4 millimeters (0.25 inch.)

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Jan. 27	63	158	95	15
Do	63	156	93	15
Do	63	160	97	15

Average expansion of bore hole, 95 cubic centimeters (5.80 cubic inches).

BITUMINITE NO. 1.

Explosive, Bituminite No. 1.
Class 4, nitroglycerin.
Manufactured by the Jefferson Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 223 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.37.
Color of explosive, light corn.
Consistency, fibrous, fine, dry, very soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, May 21, 1909.
Unit swing on this date, 2.95 inches.
Weight of charge, in grams, 310, 310, 310.
Swing, in inches, 2.82, 2.94, 2.87.
Average swing, in inches, 2.88.

$$2.88 : 2.95 :: 310 : (318).$$

Therefore the unit deflective charge of Bituminite No. 1 is 318 grams.

Gas and Dust Gallery No. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
Test 1.	*Grams.*	*Per cent.*		Test 3—Con.	*Grams.*	*Per cent.*	
May 26	318	7.91	No ignition.	May 25	318	.....	No ignition.
Do	318	8.01	Do.	Do	318	.....	Do.
Do	318	8.16	Do.	Do	318	.....	Do.
Do	318	8.16	Do.	Do	318	.....	Do.
Do	318	8.16	Do.	Do	318	.....	Do.
Do	318	8.29	Do.	Do	318	.....	Do.
Do	318	8.15	Do.	Do	318	.....	Do.
Do	318	8.13	Do.				
Do	318	8.28	Do.	Test 4.			
Do	318	8.10	Do.				
				May 24	[a] 680	4.13	No ignition.
Test 3.				Do	[a] 680	4.21	Do.
				Do	[a] 680	4.21	Do.
May 25	318	.....	No ignition.	Do	[a] 680	4.05	Do.
Do	318	.....	Do.	Do	[a] 680	4.31	Do.
Do	318	.....	Do.				

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 24	10.81	43.0	3,978
Do	11.03	43.0	3,898
Aug. 31	11.24	43.0	3,826

Average rate of detonation, 3,901 meters (12,800 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 14	21.25	26.84	9.00	0.450
Do	22.25	28.11	9.00	.450
Do	21.00	26.53	8.75	.438

Average height of flame, 27.16 inches.
Average duration of flame, 0.446 millisecond.

Impact Test.

Date, January 3, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
24	1	Explosion.	16	1	No explosion.
22	1	Do.	16	1	Explosion.
20	1	Do.	15	1	Do.
16	1	No explosion.	13	1	Do.
18	1	Explosion.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 216 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 12	6	Exploded.	Jan. 13	15	Did not explode.
Do	9	Do.	Do	14	Do.
Jan. 13	13	Do.	Do	14	Do.
Do	17	Did not explode.	Do	14	Do.

The minimum distance at which no explosion occurs established at 14 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
May 15	214	1.23	24.75	77.34	A	
May 16	214	1.23	25.75	80.47	A	78.38
Do	214	1.23	24.75	77.34	A	
May 14	214	1.23	23.25	72.66	B	
Do	214	1.23	23.25	72.66	B	72.14
Do	214	1.23	22.75	71.09	B	
Do	214	1.23	22.25	69.53	C	
Do	214	1.23	22.75	71.09	C	69.79
May 16	214	1.23	22.00	68.75	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=87.38$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.23$ $W=214$ grams.
$M=\frac{VPS}{W}=7{,}533$ kilograms per square centimeter (107,140 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, May 21, 1913.	Grams.
Solid	54.0
Liquid (water)	15.0
Gaseous	125.2

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram
	Kilograms.	*°C.*	*Calories.*
May 14	81.95	0.867	749.5
Do	81.95	.903	781.0
May 15	81.95	.903	781.0

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 770.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 19	63.50	47.50	16.00
Do	63.50	47.00	16.50
Feb. 23	63.50	47.00	16.50

Average compression, 16.3 millimeters (0.64 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 10	62	262	200	15
Do	62	261	199	15
Do	62	260	198	15

Average expansion of bore hole, 199 cubic centimeters (12.14 cubic inches).

BITUMINITE NO. 3.

Explosive, Bituminite No. 3.
Class 4, nitroglycerin.
Manufactured by the Jefferson Powder Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8¼ inches.
Average weight, 250 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.17.
Color of explosive, corn.
Consistency, fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, November 7, 1910.
Unit swing on this date, 3.16 inches.
Weight of charge, in grams, 291, 291, 291.
Swing, in inches, 3.14, 3.15, 3.11.
Average swing, in inches, 3.13.

3.13 : 3.16 : : 291 : (294).

Therefore the unit deflective charge of Bituminite No. 3 is 294 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent*	
Nov. 7	294	7.79	No ignition.	Nov. 10	294	.....	No ignition.
Do	294	7.86	Do.	Do	294	.....	Do.
Do	294	7.98	Do.	Do	294	.....	Do.
Do	294	7.91	Do.	Do	294	.....	Do.
Nov. 8	294	8.19	Do.	Do	294	.....	Do.
Do	294	7.78	Do.	Do	294	.....	Do.
Do	294	7.89	Do.	Do	294	.....	Do.
Do	294	8.12	Do.	TEST 4.			
Do	294	8.00	Do.				
Nov. 26	294	7.84	Do.	Nov. 11	[a] 680	4.02	No ignition.
				Do	[a] 680	3.98	Do.
TEST 3.				Do	[a] 680	3.92	Do.
Nov 9	294	.....	No ignition.	Do	[a] 680	4.05	Do.
Do	294	.....	Do.	Do	[a] 680	4.03	Do.
Do	294	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Nov. 14	15.32	43.5	2,839
Nov. 15	15.28	43.5	2,847
Do	15.30	43.5	2,843

Average rate of detonation, 2,843 meters (9,330 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
July 7	14.00	17.68	6.25	0.312
Do	11.00	13.89	4.75	.238
Do	9.75	12.32	4.25	.212

Average height of flame, 14.63 inches.
Average duration of flame, 0.254 millisecond.

Impact Test.

Date, June 14, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	11	1	Explosion.
18	1	Do.	10	1	No explosion.
16	1	Do.	10	1	Explosion.
14	1	Do.	9	5	No explosion.
13	1	Do.			

The maximum height at which no explosion occurs established at 9 centimeters (3.54 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 188 grams.

Date (1911).	Distance separating cartridges	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
July 18	8	Did not explode.	July 18	5	Did not explode.
Do	4	Exploded.	Do	5	Do.
Do	6	Did not explode.	Do	5	Do.

The minimum distance at which no explosion occurs established at 5 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 8	215	1.17	23.50	73.44	A	
Do	215	1.17	22.75	71.09	A	72.66
Do	215	1.17	23.50	73.44	A	
June 6	215	1.17	21.00	65.62	B	
June 7	215	1.17	21.00	65.62	B	65.36
Do	215	1.17	20.75	64.84	B	
Do	215	1.17	19.50	60.94	C	
Do	215	1.17	19.75	61.72	C	61.98
Do	215	1.17	20.25	63.28	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=84.08$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.17$. $W=215$ grams.
$M=\frac{VPS}{W}=6{,}863$ kilograms per square centimeter (97,620 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 7, 1911.	Grams.
Solid	63.8
Liquid (water)	18.7
Gaseous	121.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 14	80.95	0.811	689.9
Do	80.95	.834	709.6
Do	80.95	.842	716.4

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 705.3.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters*
July 17	63.50	49.25	14.25
Do	63.75	49.50	14.25
Do	63.50	49.50	14.00

Average compression, 14.2 millimeters (0.56 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 3	63	222	159	15
Do	63	221	158	15
Do	63	222	159	15

Average expansion of bore hole, 159 cubic centimeters (9.70 cubic inches).

BITUMINITE NO. 4.

Explosive, Bituminite No. 4.
Class 4, nitroglycerin.
Manufactured by the Jefferson Powder Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8½ inches.
Average weight, 236 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.14.
Color of explosive, drab.
Consistency, fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, November 7, 1910.
Unit swing on this date, 3.16 inches.
Weight of charge, in grams, 305, 305, 305.
Swing, in inches, 3.27, 3.16, 3.16.
Average swing, in inches, 3.20.

3.20 : 3.16 : : 305 : (301).

Therefore the unit deflective charge of Bituminite No. 4 is 301 grams.

Gas and Dust Gallery No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
Test 1.	*Grams.*	*Per cent.*		Test 3—Con.	*Grams.*	*Per cent.*	
Nov. 8	301	7.86	No ignition.	Nov. 10	301	.....	No ignition.
Do	301	7.90	Do.	Do	301	.....	Do.
Do	301	7.77	Do.	Do	301	.....	Do.
Do	301	8.00	Do.	Do	301	.....	Do.
Do	301	7.84	Do.	Do	301	.....	Do.
Do	301	7.84	Do.	Do	301	.....	Do.
Do	301	7.79	Do.	Do	301	.....	Do.
Do	301	7.93	Do.	Do	301	.....	Do.
Do	301	8.11	Do.				
Do	301	7.97	Do.	Test 4.			
Test 3.				Nov. 11	[a] 680	4.09	No ignition.
				Do	[a] 680	4.00	Do.
Nov. 10	301	.....	No ignition.	Do	[a] 680	3.89	Do.
Do	301	.....	Do.	Do	[a] 680	4.01	Do.
Do	301	.....	Do.	Do	[a] 680	4.06	Do.

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Nov. 15	18.82	43.5	2,311
Do	18.80	43.5	2,314
Do	19.17	43.5	2,269

Average rate of detonation, 2,298 meters (7,540 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
July 7	8.00	10.11	3.50	0.175
Do	10.00	12.63	5.25	.262
Do	10.25	12.95	4.75	.238

Average height of flame, 11.90 inches.
Average duration of flame, 0.225 millisecond.

Impact Test.

Date, June 15, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	13	1	Explosion.
12	1	No explosion.	12	1	Do.
16	1	Explosion.	11	3	No explosion.
14	1	Do.	11	1	Explosion.
13	2	No explosion.	10	5	No explosion.

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 184 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
July 18	5	Exploded.	July 18	6	Did not explode.
Do	7	Did not explode.	Do	6	Do.
Do	6	Do.			

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 8	215	1.14	19.00	59.37	A	58.84
Do	215	1.14	18.75	58.58	A	
Do	215	1.14	18.75	58.58	A	
Do	215	1.14	17.50	54.69	B	
June 9	215	1.14	18.00	56.25	B	55.73
Do	215	1.14	18.00	56.25	B	
June 7	215	1.14	18.00	56.25	C	
Do	215	1.14	17.75	55.47	C	55.99
June 8	215	1.14	18.00	56.25	C	

[a] Including 15 grams of the original wrapper.

P=1.911A+0.5B−1.411C=61.31 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.14. W=215 grams.
$M=\frac{VPS}{W}$=4,876 kilograms per square centimeter (69,360 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 8, 1911.	Grams.
Solid	78.2
Liquid (water)	13.5
Gaseous	119.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 15	80.95	0.736	625.3
Do	80.95	.733	622.7
July 17	80.95	.706	599.5

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 615.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 17	63.50	53.25	10.25
Do	63.50	53.75	9.75
Do	63.50	53.50	10.00

Average compression, 10.0 millimeters (0.39 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 3	63	172	109	15
Do	63	170	107	15
Do	63	168	105	15

Average expansion of bore hole, 107 cubic centimeters (6.53 cubic inches).

BITUMINITE NO. 5.

Explosive, Bituminite No. 5.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Jefferson Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 7¼ inches.
Average weight, 173 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.08.
Color of explosive, corn.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, November 7, 1910.
Unit swing on this date, 3.16 inches.
Weight of charge, in grams, 240, 240, 240.
Swing, in inches, 3.30, 3.35, 3.21.
Average swing, in inches, 3.29.

3.29 : 3.16 : : 240 : (231).

Therefore the unit deflective charge of Bituminite No. 5 is 231 grams.

Gas and Dust Gallery No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
Test 1.	*Grams.*	*Per cent.*		Test 3—Con.	*Grams.*	*Per cent.*	
Nov. 8	231	7.96	No ignition.	Nov. 10	231	.....	No ignition.
Do	231	7.98	Do.	Do	231	.....	Do.
Do	231	8.13	Do.	Nov. 11	231	.....	Do.
Do	231	7.95	Do.	Do	231	.....	Do.
Do	231	8.02	Do.	Do	231	.....	Do.
Do	231	8.03	Do.	Do	231	.....	Do.
Do	231	7.80	Do.	Do	231	.....	Do.
Do	231	7.96	Do.				
Do	231	8.13	Do.	Test 4.			
Do	231	8.00	Do.				
				Nov. 11	[a] 680	4.01	No ignition.
Test 3.				Do	[a] 680	4.04	Do.
				Do	[a] 680	3.97	Do.
Nov. 10	231	.....	No ignition.	Do	[a] 680	3.96	Do.
Do	231	.....	Do.	Nov. 14	[a] 680	3.99	Do.
Do	231	.....	Do.				

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Nov. 7	15.90	45.0	2,830
Nov. 8	16.13	45.0	2,790
Do	16.50	45.0	2,727

Average rate of detonation, 2,782 meters (9,120 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
July 8	13.25	16.74	5.50	0.275
Do	13.00	16.42	4.75	.238
Do	11.75	14.84	4.00	.200

Average height of flame, 16.00 inches.
Average duration of flame, 0.238 millisecond.

Impact Test.

Date, June 15, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
16	1	No explosion.	16	1	Explosion.
20	1	Explosion.	15	1	Do.
18	1	Do.	14	4	No explosion.
14	1	No explosion.			

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 174 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
July 19	2	Exploded.	July 19	4	Did not explode.
Do	4	Did not explode.	Do	4	Do.
Do	3	Exploded.			

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 14	214	1.08	31.50	98.44	A	97.66
Do	214	1.08	31.25	97.66	A	
Do	214	1.08	31.00	96.88	A	
June 9	214	1.08	29.75	92.97	B	92.71
Do	214	1.08	30.25	94.53	B	
June 21	214	1.08	29.00	90.62	B	
June 10	214	1.08	28.25	88.28	C	88.54
Do	214	1.08	28.00	87.50	C	
June 20	214	1.08	28.75	89.84	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=108.05$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.08$. $W=214$ grams.
$M=\frac{VPS}{W}=8{,}179$ kilograms per square centimeter (116,340 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 21 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 9, 1911.	Grams.
Solid	14.3
Liquid (water)	81.0
Gaseous	112.9

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*° C.*	*Calories.*
July 18	80.95	1.336	1,146.9
Do	80.95	1.307	1,121.9
Do	80.95	1.342	1,152.1

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,140.3.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 11	63.25	51.00	12.25
Do	63.50	51.00	12.50
Do	63.50	51.00	12.50

Average compression, 12.4 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 3	63	226	163	15
Do	63	220	157	15
Do	63	220	157	15

Average expansion of bore hole, 159 cubic centimeters (9.70 cubic inches).

BLACK DIAMOND NO. 2-A.

Explosive, Black Diamond No. 2-A.
Class 4, nitroglycerin.
Manufactured by the Illinois Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 196 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.17.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, January 29, 1912.
Unit swing on this date, 3.62 inches.
Weight of charge, in grams, 280, 280, 280.
Swing, in inches, 3.63, 3.66, 3.53.
Average swing, in inches, 3.61.

3.61 : 3.62 : : 280 : (281).

Therefore the unit deflective charge of Black Diamond No. 2-A is 281 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 31	281	7.87	No ignition.	Feb. 5	281	.....	No ignition.
Do	281	8.28	Do.	Do	281	.....	Do.
Do	281	8.69	Do.	Do	281	.....	Do.
Do	281	8.00	Do.	Do	281	.....	Do.
Do	281	8.22	Do.	Do	281	.....	Do.
Do	281	8.49	Do.	Do	281	.....	Do.
Do	281	7.88	Do.	Feb. 6	281	.....	Do.
Do	281	7.99	Do.				
Do	281	8.19	Do.	TEST 4.			
Do	281	7.95	Do.				
				Feb. 2	[a] 680	4.24	No ignition.
TEST 3.				Do	[a] 680	4.10	Do.
				Do	[a] 680	3.95	Do.
Feb. 5	281	.....	No ignition.	Do	[a] 680	3.88	Do.
Do	281	.....	Do.	Do	[a] 680	4.02	Do.
Do	281	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 30	11.50	44.5	3,819
Feb. 5	11.70	45.0	3,846
Feb. 6	11.40	44.0	3,860

Average rate of detonation, 3,842 meters (12,600 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 30	12.75	16.11	5.00	0.250
Do	13.00	16.42	4.75	.238
June 4	8.00	10.11	1.75	.088

Average height of flame, 14.21 inches.
Average duration of flame, 0.192 millisecond.

Impact Test.

Date, April 19, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	1	No explosion.
18	1	Do.	17	5	Do.

The maximum height at which no explosion occurs established at 17 centimeters (6.69 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 188 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 1	3	Exploded.	May 1	6	Exploded.
Do	5	Do.	Do	7	Did not explode.
Do	9	Did not explode.	Do	7	Do.
Do	7	Do.			

The minimum distance at which no explosion occurs established at 7 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 18	213	1.17	20.25	63.28	A	
Apr. 19	213	1.17	20.50	64.06	A	63.80
Do	213	1.17	20.50	64.06	A	
Apr. 22	213	1.17	19.50	60.94	B	
Do	213	1.17	20.25	63.28	B	62.76
Do	213	1.17	20.50	64.06	B	
Apr. 19	213	1.17	18.25	57.03	C	
Do	213	1.17	18.25	57.03	C	57.81
Do	213	1.17	19.00	59.38	C	

[a] Including 13 grams of the original wrapper.

P=1.911A+0.5B−1.411C=71.73 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.17. W=213 grams.
$M=\frac{VPS}{W}$=5,910 kilograms per square centimeter (84,060 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 29, 1912.	Grams.
Solid	66.9
Liquid (water)	13.0
Gaseous	124.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Apr. 8	81.95	0.871	756.6
Apr. 9	81.95	.842	731.2
Apr. 13	81.95	.849	737.3

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 741.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 17	63.25	46.25	17.00
Do	63.00	46.00	17.00
Do	63.25	46.25	17.00

Average compression, 17.0 millimeters (0.67 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 15	63	220	157	19
Do	63	226	163	19
Do	63	228	165	19

Average expansion of bore hole, 162 cubic centimeters (9.88 cubic inches).

BLACK DIAMOND NO. 3-A.

Explosive, Black Diamond No. 3-A.
Class 4, nitroglycerin.
Manufactured by the Illinois Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 183 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.09.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, January 29, 1912.
Unit swing on this date, 3.62 inches.
Weight of charge, in grams, 290, 290, 290.
Swing, in inches, 3.63, 3.54, 3.55.
Average swing, in inches, 3.57.

3.57 : 3.62 : : 290 : (294).

Therefore the unit deflective charge of Black Diamond No. 3-A is 294 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 31	294	7.81	No ignition.	Feb. 6	294	.....	No ignition.
Do	294	7.91	Do.	Do	294	.....	Do.
Do	294	8.29	Do.	Do	294	.....	Do.
Do	294	8.41	Do.	Do	294	.....	Do.
Feb. 1	294	8.39	Do.	Do	294	.....	Do.
Do	294	8.12	Do.	Do	294	.....	Do.
Do	294	8.31	Do.	Do	294	.....	Do.
Do	294	8.72	Do.				
Do	294	7.82	Do.	TEST 4.			
Do	294	8.28	Do.				
				Feb. 5	[a] 680	3.79	No ignition.
TEST 3.				Do	[a] 680	3.88	Do.
				Do	[a] 680	3.98	Do.
Feb. 6	294	.....	No ignition.	Do	[a] 680	4.02	Do.
Do	294	.....	Do.	Do	[a] 680	3.91	Do.
Do	294	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 31	12.85	44.0	3,424
Feb. 5	13.10	44.0	3,359
Feb. 6	12.85	44.0	3,423

Average rate of detonation, 3,402 meters (11,160 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 30	16.00	20.21	6.00	0.300
Do	13.00	16.42	6.50	.325
Do	13.00	16.42	4.25	.213

Average height of flame, 17.68 inches.
Average duration of flame, 0.279 millisecond.

Impact Test.

Date, April 19, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	24	1	Explosion.
25	1	Explosion.	22	1	Do.
24	1	No explosion.	21	5	No explosion.

The maximum height at which no explosion occurs established at 21 centimeters (8.27 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 175 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 2	7	Did not explode.	May 2	4	Did not explode.
Do	3	Exploded.	Do	4	Do.
Do	5	Did not explode.	Do	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 22	213	1.09	21.00	65.62	A	
May 2	213	1.09	22.00	68.75	A	67.71
Do	213	1.09	22.00	68.75	A	
Apr. 23	213	1.09	20.50	64.06	B	
Do	213	1.09	19.50	60.94	B	63.02
Apr. 24	213	1.09	20.50	64.06	B	
Do	213	1.09	19.25	60.16	C	
Do	213	1.09	19.25	60.16	C	60.42
Do	213	1.09	19.50	60.94	C	

[a] Including 13 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=75.65$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.09$. $W=213$ grams.
$M=\frac{VPS}{W}=5{,}807$ kilograms per square centimeter (82,590 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 30, 1912.	Grams.
Solid	75.0
Liquid (water)	11.5
Gaseous	119.2

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams. [a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	°*C.*	*Calories.*
Apr. 12	81.95	0.884	768.0
Do	81.95	.850	738.2
Apr. 15	81.95	.887	770.6

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 758.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 17	63.25	48.50	14.75
Do	63.25	48.75	14.50
Do	63.25	49.00	14.25

Average compression, 14.5 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	°*C.*
Apr. 15	63	228	165	19
Do	63	231	168	19
Do	63	228	165	19

Average expansion of bore hole, 166 cubic centimeters (10.13 cubic inches).

BLACK DIAMOND NO. 5.

Explosive, Black Diamond No. 5.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Illinois Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 179 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.10.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, January 29, 1912.
Unit swing on this date, 3.62 inches.
Weight of charge, in grams, 300, 300, 300.
Swing, in inches, 3.71, 3.82, 3.77.
Average swing, in inches, 3.77.

3.77 : 3.62 : : 300 : (288).

Therefore the unit deflective charge of Black Diamond No. 5 is 288 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 1	288	8.13	No ignition.	Feb. 6	288	.....	No ignition.
Do	288	8.22	Do.	Do	288	.....	Do.
Do	288	7.99	Do.	Do	288	.....	Do.
Do	288	8.17	Do.	Do	288	.....	Do.
Do	288	8.27	Do.	Do	288	.....	Do.
Feb. 2	288	8.23	Do.	Do	288	.....	Do.
Do	288	8.16	Do.	Do	288	.....	Do.
Do	288	7.97	Do.				
Do	288	8.36	Do.	TEST 4.			
Do	288	8.25	Do.				
				Feb. 2	[a] 680	4.05	No ignition.
TEST 3.				Feb. 3	[a] 680	4.00	Do.
				Do	[a] 680	4.11	Do.
Feb. 6	288	.....	No ignition.	Do	[a] 680	4.07	Do.
Do	288	.....	Do.	Do	[a] 680	4.21	Do.
Do	288	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 30	24.10	44.5	1,847
Feb. 5	24.45	44.5	1,820
Feb. 6	24.20	44.5	1,839

Average rate of detonation, 1,835 meters (6,020 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
May 16	7.00	8.84	2.50	0.125
Do	8.25	10.42	4.00	.200
Do	3.50	4.42	1.50	.075

Average height of flame, 7.89 inches.
Average duration of flame, 0.133 millisecond.

Impact Test.

Date, March 5, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	17	2	No explosion.
20	1	Explosion.	17	1	Explosion.
18	3	No explosion.	16	5	No explosion.
18	1	Explosion.			

The maximum height at which no explosion occurs established at 16 centimeters (6.30 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 177 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 2	7	Did not explode.	May 2	2	Did not explode.
Do	3	Do.	Do	2	Do.
Do	1	Exploded.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 25	213	1.10	30.25	94.53	A	
Do	213	1.10	29.50	92.19	A	93.23
Do	213	1.10	29.75	92.97	A	
Do	213	1.10	25.75	80.47	B	
Do	213	1.10	26.75	83.59	B	81.77
Do	213	1.10	26.00	81.25	B	
Apr. 24	213	1.10	23.75	74.22	C	
Do	213	1.10	23.75	74.22	C	73.44
Do	213	1.10	23.00	71.88	C	

[a] Including 13 grams of the original wrapper.

P=1.911A+0.5B−1.411C=115.42 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.10. W=213 grams.
$M=\frac{VPS}{W}$=8,941 kilograms per square centimeter (127,170 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 30, 1912.	Grams.
Solid	33.7
Liquid (water)	61.5
Gaseous	106.2

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 17	63.25	53.50	9.75
Do	63.25	53.75	9.50
Do	63.25	53.75	9.50

Average compression, 9.6 millimeters (0.38 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 25	62	206	144	19
Do	62	200	138	19
Do	62	204	142	19

Average expansion of bore hole, 141 cubic centimeters (8.60 cubic inches).

CAMERON MINE POWDER NO. 1-A.

Explosive, Cameron mine powder No. 1-A.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Cameron Powder Manufacturing Co
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 148 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.90.
Color of explosive, corn.
Consistency, granular and fibrous, fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, March 7, 1911.

Unit swing on this date, 3.13 inches.

Weight of charge, in grams, 220, 220, 220.

Swing, in inches, 3.04, 3.11, 2.97.

Average swing, in inches, 3.04.

3.04 : 3.13 : : 220 : (227).

Therefore the unit deflective charge of Cameron mine powder No. 1-A is 227 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Mar. 8	227	7.94	No ignition.	Mar. 14	227	.....	No ignition.
Do	227	7.80	Do.	Do	227	.....	Do.
Mar. 9	227	7.83	Do.	Do	227	.....	Do.
Do	227	7.93	Do.	Do	227	.....	Do.
Do	227	7.93	Do.	Do	227	.....	Do.
Do	227	8.25	Do.	Do	227	.....	Do.
Do	227	7.78	Do.	Do	227	.....	Do.
Do	227	8.11	Do.	TEST 4.			
Do	227	8.14	Do.				
Mar. 10	227	7.93	Do.	Mar. 8	[a] 680	3.93	No ignition.
TEST 3.				Mar. 20	[a] 680	4.07	Do.
				Do	[a] 680	4.14	Do.
Mar. 14	227	.....	No ignition.	Do	[a] 680	3.93	Do.
Do	227	.....	Do.	Do	[a] 680	3.97	Do.
Do	227	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 7	13.56	45.0	3,319
Do	13.30	45.0	3,383
Do	13.58	45.0	3,314

Average rate of detonation, 3,339 meters (10,950 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 28	7.00	8.84	2.00	0.100
Do	7.50	9.47	2.00	.100
Do	8.75	11.05	3.00	.150

Average height of flame, 9.79 inches.

Average duration of flame, 0.117 millisecond.

Impact Test.

Date, July 28, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	12	1	Explosion.
14	1	Do.	10	1	Do.
8	1	No explosion.	9	5	No explosion.
10	1	Do.			

The maximum height at which no explosion occurs established at 9 centimeters (3.54 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 152 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Oct. 2	4	Did not explode.	Oct. 2	3	Did not explode.
Do	3	Do.	Do	3	Do.
Do	2	Exploded.			

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 4	215	0.96	31.00	96.88	A	98.18
Oct. 5	215	.96	32.00	100.00	A	
Do	215	.96	31.25	97.66	A	
Oct. 6	215	.94	31.25	97.66	B	94.79
Do	215	.94	30.00	93.75	B	
Oct. 7	215	.94	29.75	92.97	B	
Do	215	.94	28.75	89.84	C	90.62
Oct. 9	215	.94	28.75	89.84	C	
Do	215	.94	29.50	92.19	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=107.15$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.90$. $W=215.0$ grams.

$M=\frac{VPS}{W}=6{,}729$ kilograms per square centimeter (95,710 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 15 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, June 23, 1911.

	Grams.
Solid	9.6
Liquid (water)	66.5
Gaseous	129.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.3 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 29	81.95	1.267	1,096.0
Jan. 30	81.95	1.227	1,061.1
Do	81.95	1.248	1,079.5

[a] Including 7.3 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,078.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 23	63.25	46.00	17.25
Do	63.50	46.00	17.50
Do	63.50	46.50	17.00

Average compression, 17.2 millimeters (0.68 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCK.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 11	63	371	308	15
Do	63	376	313	15
Do	63	383	320	15

Average expansion of bore hole, 314 cubic centimeters (19.15 cubic inches).

CAMERON MINE POWDER NO. 2-A.

Explosive, Cameron mine powder No. 2-A.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Cameron Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 157 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.94.
Color of explosive, gray.
Consistency, granular and fibrous, fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, March 7, 1911.
Unit swing on this date, 3.13 inches.
Weight of charge, in grams, 220, 220, 220.
Swing, in inches, 3.05, 2.98, 2.97.
Average swing, in inches, 3.00.

3.00 : 3.13 : : 220 : (230).

Therefore the unit deflective charge of Cameron mine powder No. 2–A is 230 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Mar. 9	230	8.04	No ignition.	Mar. 14	230	.....	No ignition.
Do	230	8.21	Do.	Do	230	.....	Do.
Do	230	7.82	Do.	Do	230	.....	Do.
Do	230	7.99	Do	Do	230	.....	Do.
Do	230	8.03	Do.	Do	230	.....	Do.
Do	230	7.87	Do.	Do	230	.....	Do.
Do	230	8.06	Do.	Do	230	.....	Do.
Mar. 10	230	7.86	Do.				
Do	230	8.05	Do.	TEST 4.			
Do	230	8.06	Do.				
				Mar. 16	[a] 680	3.97	No ignition.
TEST 3.				Do	[a] 680	3.82	Do.
				Do	[a] 680	3.88	Do.
Mar. 14	230	.....	No ignition.	Do	[a] 680	3.99	Do.
Do	230	.....	Do.	Do	[a] 680	3.94	Do.
Do	230	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 23	14.32	45.0	3,142
Mar. 31	14.18	45.0	3,173
Do	13.63	45.0	3,302

Average rate of detonation, 3,206 meters (10,520 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911–12).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 28	10.75	13.58	3.50	0.175
Do	7.50	9.47	2.00	.100
Jan. 3	16.25	20.53	6.50	.325

Average height of flame, 14.53 inches.
Average duration of flame, 0.200 millisecond.

IMPACT TEST.

Date, July 29, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	Explosion.	15	1	Explosion.
20	1	Do.	14	1	Do.
16	1	No explosion.	13	1	Do.
18	1	Explosion.	12	5	No explosion.
16	1	Do.			

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 160 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Oct. 2	3	Did not explode.	Oct. 3	½	Exploded.
Do	2	Do.	Do	1	Did not explode.
Do	1	Do.	Do	1	Do.
Do	0	Exploded.			

The minimum distance at which no explosion occurs established at 1 inch.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 10	214	0.94	30.75	96.09	A	
Oct. 11	214	.94	30.00	93.75	A	96.09
Do	214	.94	31.50	98.44	A	
Oct. 10	214	.94	28.00	87.50	B	
Do	214	.94	27.75	86.72	B	87.24
Do	214	.94	28.00	87.50	B	
Oct. 9	214	.94	27.50	85.94	C	
Do	214	.94	27.00	84.38	C	86.20
Oct. 10	214	.94	28.25	88.28	C	

[a] Including 14 grams of the original wrapper.

P=1.911A+0.5B−1.411C=105.62 kilograms per square centimeter.

V=15,000 cubic centimeters. S=0.94. W=214 grams.

$M=\frac{VPS}{W}$=6,959 kilograms per square centimeter (98,990 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 23, 1911.

	Grams.
Solid	12.4
Liquid (water)	63.5
Gaseous	130.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 19	81.95	1.156	1,002.1
Do	81.95	1.170	1,014.4
Jan. 20	81.95	1.147	994.2

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,003.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 23	63.50	51.25	12.25
Do	63.75	51.50	12.25
Do	63.50	51.50	12.00

Average compression, 12.2 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCK.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 11	63	362	299	15
Do	63	355	292	15
Do	63	357	294	15

Average expansion of bore hole, 295 cubic centimeters (18 cubic inches).

CAMERON MINE POWDER NO. 2-A, L. F.

Explosive, Cameron mine powder No. 2-A, L. F.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Cameron Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 183 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.18.
Color of explosive, corn.
Consistency, granular, very fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 25, 1911.

Unit swing on this date, 3.41 inches.

Weight of charge, in grams, 235, 235, 235.

Swing, in inches, 3.47, 3.43, 3.42.

Average swing, in inches, 3.44.

3.44 : 3.41 : : 235 : (233).

Therefore the unit deflective charge of Cameron mine powder No. 2-A, L. F., is 233 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 27	233	8.23	No ignition.	Sept. 29	233	.....	No ignition.
Do	233	8.15	Do.	Sept. 30	233	.....	Do.
Do	233	8.30	Do.	Do	233	.....	Do.
Do	233	8.35	Do.	Do	233	.....	Do.
Do	233	8.03	Do.	Oct. 2	233	.....	Do.
Do	233	8.34	Do.	Do	233	.....	Do.
Sept. 28	233	8.09	Do.	Do	233	.....	Do.
Do	233	7.90	Do.				
Do	233	8.12	Do.	TEST 4.			
Do	233	8.23	Do.				
				Sept. 28	[a] 680	4.25	No ignition.
TEST 3.				Do	[a] 680	4.28	Do.
				Do	[a] 680	4.11	Do.
Sept. 29	233	.....	No ignition.	Do	[a] 680	4.09	Do.
Do	233	.....	Do.	Do	[a] 680	4.06	Do.
Do	233	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 26	13.42	45.0	3,353
Sept. 27	13.21	45.0	3,407
Do	13.83	45.0	3,254

Average rate of detonation, 3,338 meters (10,950 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 28	9.75	12.32	5.00	0.250
Do	5.75	7.26	2.75	.138
Do	9.75	12.32	5.75	.288

Average height of flame, 10.63 inches.

Average duration of flame, 0.225 millisecond.

Impact Test.

Date, November 9, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
8	1	No explosion.	28	1	No explosion.
16	1	Do.	30	1	Do.
32	1	Explosion.	31	1	Explosion.
24	1	No explosion.	30	4	No explosion.

The maximum height at which no explosion occurs established at 30 centimeters (11.81 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 179 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 12	6	Did not explode.	Dec. 12	3	Did not explode.
Do	4	Do.	Do	3	Do.
Do	2	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.40 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 29	106.5	1.11	20.75	51.88	A	
Jan. 30	106.5	1.11	21.50	53.75	A	53.13
Do	106.5	1.11	21.50	53.75	A	
Do	106.5	1.11	19.00	47.50	B	
Do	106.5	1.11	19.00	47.50	B	48.13
Do	106.5	1.11	19.75	49.38	B	
Do	106.5	1.11	19.25	48.12	C	
Do	106.5	1.11	19.00	47.50	C	47.91
Do	106.5	1.11	19.25	48.12	C	

[a] Including 6.5 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=58.00$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.11$. $W=106.5$ grams.

$M=\frac{VPS}{W}=9{,}068$ kilograms per square centimeter (128,980 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 13 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

December 9, 1912.	Grams.
Solid	18.7
Liquid (water)	73.0
Gaseous	105.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 20	81.95	1.369	1,193.8
Do	81.95	1.364	1,189.4
Feb. 21	81.95	1.330	1,159.6

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,180.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 27	63.25	50.00	13.25
Do	63.50	50.25	13.25
Do	63.75	50.00	13.75

Average compression, 13.4 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCK.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 15	63	266	203	15
Do	63	260	197	15
Do	63	270	207	15

Average expansion of bore hole, 202 cubic centimeters (12.32 cubic inches).

CAMERON MINE POWDER NO. 3-A.

Explosive, Cameron mine powder No. 3-A.
Class 4, nitroglycerin.
Manufactured by the Cameron Powder Manufacturing Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 176 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, corn.
Consistency, fibrous, fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, March 7, 1911.

Unit swing on this date, 3.13 inches.

Weight of charge, in grams, 310, 310, 310.

Swing, in inches, 3.24, 3.28, 3.32.

Average swing, in inches, 3.28.

3.28 : 3.13 : : 310 : (296).

Therefore the unit deflective charge of Cameron mine powder No. 3-A is 296 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Mar. 11	296	7.99	No ignition.	Mar. 15	296	.....	No ignition.
Do	296	8.06	Do.	Do	296	.....	Do.
Do	396	7.79	Do.	Do	296	.....	Do.
Do	296	8.07	Do.	Do	296	.....	Do.
Do	296	7.98	Do.	Do	296	.....	Do.
Do	296	8.11	Do.	Do	296	.....	Do.
Do	296	8.08	Do.	Do	296	.....	Do.
Mar. 13	296	7.87	Do.				
Do	296	7.80	Do.	TEST 4.			
Do	296	8.02	Do.				
				Mar. 18	a 680	4.15	No ignition.
TEST 3.				Do	a 680	4.01	Do.
				Mar. 20	a 680	4.06	Do.
Mar. 15	296	.....	No ignition.	Do	a 680	4.03	Do.
Do	296	.....	Do.	Do	a 690	4.03	Do.
Do	296	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 8	13.73	45.0	3,277
Mar. 10	13.28	45.0	3,389
Do	13.50	45.0	3,333

Average rate of detonation, 3,333 meters (10,930 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 3	15.50	19.58	6.75	0.338
Do	17.50	22.11	7.50	.375
Do	17.75	22.42	7.00	.350

Average height of flame, 21.37 inches.

Average duration of flame, 0.354 millisecond.

IMPACT TEST.

Date, July 24, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	13	1	Explosion.
16	1	Do.	12	1	No explosion.
12	1	No explosion.	12	1	Explosion.
15	1	Explosion.	11	5	No explosion.
14	1	Do.			

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 175 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Oct. 3	4	Did not explode.	Oct. 3	5	Did not explode.
Do	2	Exploded.	Do	5	Do.
Do	3	Do.	Do	5	Do.
Do	4	Do.			

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 11	213	1.03	21.25	66.41	A	
Do	213	1.03	21.25	66.41	A	65.37
Oct. 12	213	1.03	20.25	63.28	A	
Do	213	1.03	20.75	64.84	B	
Do	213	1.03	21.25	66.41	B	66.15
Oct. 13	213	1.03	21.50	67.19	B	
Do	213	1.03	21.25	66.41	C	
Do	213	1.03	21.25	66.41	C	65.63
Oct. 14	213	1.03	20.50	64.06	C	

[a] Including 13 grams of the original wrapper.

P=1.911A+0.5B−1.411C=65.39 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.03. W=213 grams.

$M=\frac{VPS}{W}$=4,743 kilograms per square centimeter (67,470 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 26, 1911.

	Grams.
Solid	59.2
Liquid (water)	12.1
Gaseous	127.1

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 27	81.95	0.753	653.1
Dec. 29	81.95	.772	669.7
Do	81.95	.772	669.7

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 664.2.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 26	63.50	49.00	14.50
Do	63.50	48.75	14.75
Do	63.50	49.00	14.50

Average compression, 14.6 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCK.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 11	63	240	177	15
Do	63	240	177	15
Do	63	246	183	15

Average expansion of bore hole, 179 cubic centimeters (10.92 cubic inches).

CARBONITE NO. 4.

Explosive, Carbonite No. 4.
Class 4, nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 246 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.05.
Color of explosive, drab.
Consistency, granular; coarse; dry; hard; slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 11, 1910.

Unit swing on this date, 3.26 inches.

Weight of charge, in grams, 320, 320, 320.

Swing, in inches, 3.20, 3.18, 3.20.

Average swing, in inches, 3.19.

3.19 : 3.26 : : 320 : (327).

Therefore the unit deflective charge of Carbonite No. 4 is 327 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Feb. 12	327	8.16	No ignition.	Feb. 16	327	.....	No ignition.
Do	327	8.16	Do.	Do	327	.....	Do.
Do	327	8.26	Do.	Do	327	.....	Do.
Do	327	8.26	Do.	Feb. 17	327	.....	Do.
Do	327	7.87	Do.	Do	327	.....	Do.
Do	327	8.21	Do.	Do	327	.....	Do.
Feb. 14	327	7.90	Do.	Do	327	.....	Do.
Do	327	8.23	Do.				
Do	327	8.22	Do.	TEST 4.			
Do	327	8.13	Do.				
				Feb. 28	[a] 680	4.15	No ignition.
TEST 3.				Do	[a] 680	3.97	Do.
				Do	[a] 680	4.10	Do.
Feb. 16	327	.....	No ignition.	Do	[a] 680	4.10	Do.
Do	327	.....	Do.	Do	[a] 680	4.10	Do.
Do	327	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 9	18.33	43.0	2,346
Feb. 15	18.31	43.0	2,348
Mar. 2	18.50	43.0	2,324

Average rate of detonation, 2,339 meters (7,670 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 29	7.00	8.84	2.50	0.125
Do	7.25	9.16	2.00	.100
Do	4.00	5.05	1.50	.075

Average height of flame, 7.68 inches.

Average duration of flame 0.100 millisecond.

Impact Test.

Date, December 5, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
8	1	No explosion.	9	3	No explosion.
16	1	Explosion.	9	1	Explosion.
12	1	Do.	8	1	Do.
10	1	No explosion.	7	1	No explosion.
11	1	Explosion.	7	1	Explosion.
10	1	Do.	6	5	No explosion.

The maximum height at which no explosion occurs established at 6 centimeters (2.36 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 169 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 5	8	Exploded.	Dec. 8	9	Did not explode.
Do	12	Did not explode.	Do	9	Do.
Dec. 8	10	Do.	Do	9	Do.

The minimum distance at which no explosion occurs established at 9 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 17	210	1.05	23.25	58.12	A	
Do	210	1.05	22.50	56.25	A	57.29
Do	210	1.05	23.00	57.50	A	
Oct. 18	210	1.05	22.50	56.25	B	
Do	210	1.05	22.00	55.00	B	55.00
Do	210	1.05	21.50	53.75	B	
Do	210	1.05	20.50	51.25	C	
Oct. 19	210	1.05	20.50	51.25	C	52.08
Do	210	1.05	21.50	53.75	C	

[a] Including 10 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=63.50$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.05$. $W=210$ grams.

$M=\frac{VPS}{W}=4{,}762$ kilograms per square centimeter (67,740 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 13 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, September 6, 1910.	Grams.
Solid	76.2
Liquid (water)	20.0
Gaseous	113.7

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 13	81.95	0.695	602.1
Jan. 14	81.95	.706	611.7
Do	81.95	.695	602.1

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 605.3.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 27	63.75	55.50	8.25
Do	63.50	55.00	8.50
Do	63.00	54.75	8.25

Average compression, 8.3 millimeters (0.33 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 15	63	170	107	15
Do	63	175	112	15
Do	63	170	107	15

Average expansion of bore hole, 109 cubic centimeters (6.65 inches).

CARBONITE NO. 5.

Explosive, Carbonite No. 5.
Class 4, nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, $\frac{7}{8}$ inch.
Length of cartridge, 8 inches.
Average weight, 86 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.06.
Color of explosive, buff.
Consistency, fibrous, fine, very dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, January 23, 1911.

Unit swing on this date, 3.05 inches.

Weight of charge, in grams, 320, 320, 320.

Swing, in inches, 3.26, 3.20, 3.18.

Average swing, in inches, 3.21.

3.21 : 3.05 :: 320 : (304).

Therefore the unit deflective charge of Carbonite No. 5 is 304 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 25	304	7.85	No ignition.	Feb. 2	304	.....	No ignition.
Do	304	7.93	Do.	Do	304	.....	Do.
Do	304	7.77	Do.	Do	304	.....	Do.
Do	304	8.13	Do.	Do	304	.....	Do.
Do	304	7.79	Do.	Do	304	.....	Do.
Do	304	8.22	Do.	Do	304	.....	Do.
Do	304	8.13	Do.	Do	304	.....	Do.
Do	304	8.14	Do.				
Do	304	8.13	Do.	TEST 4.			
Do	304	8.08	Do.				
				Jan. 30	a 680	4.04	No ignition.
TEST 3.				Do	a 680	3.94	Do.
				Do	a 680	3.92	Do.
Feb. 2	304	.....	No ignition.	Do	a 680	4.13	Do.
Do	304	.....	Do.	Do	a 680	4.06	Do.
Do	304	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 31	14.16	45.0	3,178
Do	14.82	45.0	3,088
Do	14.70	45.0	3,061

Average rate of detonation, 3,092 meters (10,140 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters*	*Milliseconds.*
July 27	13.00	16.42	6.50	0.325
Do	13.00	16.42	5.75	.288
Do	9.75	12.32	5.00	.250

Average height of flame, 15.05 inches.

Average duration of flame, 0.288 millisecond.

IMPACT TEST.

Date, July 26, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	14	1	No explosion.
14	1	No explosion.	14	1	Explosion.
17	1	Explosion.	13	1	Do.
16	1	Do.	12	1	Do.
15	1	Do.	11	5	No explosion.

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 180.5 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 21.....	8	Did not explode.	Sept. 21.....	4	Exploded.
Do.....	5	Do.	Do.....	5	Did not explode.
Do.....	3	Exploded.	Do.....	5	Do.

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
July 25.....	221	1.06	25.50	63.75	A	
Do.....	221	1.06	26.00	65.00	A	65.00
Do.....	221	1.06	26.50	66.25	A	
July 24.....	221	1.06	25.25	63.13	B	
Do.....	221	1.06	25.25	63.13	B	62.92
Do.....	221	1.06	25.00	62.50	B	
Do.....	221	1.06	25.00	62.50	C	
Do.....	221	1.06	25.00	62.50	C	61.67
Do.....	221	1.06	24.00	60.00	C	

[a] Including 21 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=68.66$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.06$. $W=221$ grams.

$M=\frac{VPS}{W}=4{,}940$ kilograms per square centimeter (70,270 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 21 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 15, 1911.

	Grams.
Solid	64.1
Liquid (water)	9.3
Gaseous	139.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 24	80.95	0.685	565.6
July 26	80.95	.720	594.9
Do	80.95	.720	594.9

[a] Including 10.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 585.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 31	63.50	51.75	11.75
Do	63.50	52.25	11.25
Do	63.50	51.50	12.00

Average compression, 11.7 millimeters (0.46 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 7	63	217	154	15
Do	63	222	159	15
Do	63	218	155	15

Average expansion of bore hole, 156 cubic centimeters (9.52 cubic inches).

CARBONITE NO. 6.

Explosive, Carbonite No. 6.
Class 4, nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, $\frac{1}{4}$ inch.
Length of cartridge, 8 inches.
Average weight, 86 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.05.
Color of explosive, corn.
Consistency, granular, fine, very dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date January 23, 1911.

Unit swing on this date, 3.05 inches.

Weight of charge, in grams, 340, 340, 340.

Swing, in inches, 3.06, 2.95, 3.03.

Average swing, in inches, 3.01.

3.01 : 3.05 : : 340 : (345).

Therefore the unit deflective charge of Carbonite No. 6 is 345 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 25	345	8.23	No ignition.	Feb. 3	345	.....	No ignition.
Jan. 26	345	7.91	Do.	Do	345	.....	Do.
Do	345	7.93	Do.	Do	345	.....	Do.
Do	345	7.84	Do.	Do	345	.....	Do.
Do	345	8.19	Do.	Do	345	.....	Do.
Do	345	7.98	Do.	Do	345	.....	Do.
Do	345	7.90	Do.	Do	345	.....	Do.
Do	345	8.06	Do.				
Do	345	8.03	Do.	TEST 4.			
Do	345	7.99	Do.				
				Jan. 30	[a] 680	3.99	No ignition.
TEST 3.				Jan. 31	[a] 680	3.94	Do.
				Do	[a] 680	4.02	Do.
Feb. 3	345	.....	No ignition.	Do	[a] 680	4.02	Do.
Do	345	.....	Do.	Do	[a] 680	4.12	Do.
Do	345	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 31	19.55	45.0	2,302
Do	19.38	45.0	2,322
Feb. 10	20.18	45.0	2,230

Average rate of detonation, 2,285 meters (7,490 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
July 27	9.75	12.32	4.50	0.225
Do	11.50	14.53	7.00	.350
Do	11.25	14.21	6.00	.300

Average height of flame, 13.69 inches.

Average duration of flame, 0.292 millisecond.

Impact Test.

Date, July 26, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	15	1	Explosion.
20	1	Explosion.	14	1	Do.
17	1	Do.	13	1	Do.
16	1	Do.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 179 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 22	7	Did not explode.	Sept. 22	3	Exploded.
Do	4	Do.	Do	4	Did not explode.
Do	3	Do.	Do	4	Do.
Do	2	Exploded.			

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
July 26	221	1.05	22.00	55.00	A	
Do	221	1.05	21.75	54.38	A	55.42
Do	221	1.05	22.75	56.88	A	
July 25	221	1.05	21.50	53.75	B	
Do	221	1.05	20.75	51.88	B	53.13
Do	221	1.05	21.50	53.75	B	
Do	221	1.05	20.25	50.63	C	
Do	221	1.05	20.00	50.00	C	50.21
Do	221	1.05	20.00	50.00	C	

[a] Including 21 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=61.63$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.05$. $W=221$ grams.

$M=\frac{VPS}{W}=4{,}392$ kilograms per square centimeter (62,470 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 21 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, June 16, 1911.

	Grams.
Solid	71.7
Liquid (water)	11.8
Gaseous	123.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 27	80.95	0.678	559.8
Do	80.95	.706	583.2
Do	80.95	.697	575.6

[a] Including 10.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 572.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 31	63.25	54.25	9.00
Do	63.50	54.75	8.75
Do	63.50	54.75	8.75

Average compression, 8.8 millimeters (0.35 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 7	63	183	120	15
Do	63	181	118	15
Do	63	188	125	15

Average expansion of bore hole, 121 cubic centimeters (7.38 cubic inches).

COALITE A.

Explosive, Coalite A.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 171 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.98.
Color of explosive, ecru.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, January 10, 1912.
Unit swing on this date, 3.40 inches.
Weight of charge, in grams, 245, 245, 245.
Swing, in inches, 3.36, 3.38, 3.44.
Average swing, in inches, 3.39.

3.39 : 3.40 : : 245 : (246).

Therefore the unit deflective charge of Coalite A is 246 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 15	246	8.24	No ignition.	Jan. 23	246	.....	No ignition.
Do	246	8.41	Do.	Do	246	.....	Do.
Do	246	8.09	Do.	Do	246	.....	Do.
Do	246	7.85	Do.	Do	246	.....	Do.
Jan. 16	246	7.98	Do.	Do	246	.....	Do.
Do	246	7.92	Do.	Do	246	.....	Do.
Do	246	7.88	Do.	Do	246	.....	Do.
Do	246	8.32	Do.				
Do	246	8.04	Do.	TEST 4.			
Do	246	8.16	Do.				
				Jan. 19	[a] 680	4.07	No ignition.
TEST 3.				Jan. 20	[a] 680	3.91	Do.
				Do	[a] 680	3.94	Do.
Jan. 23	246	.....	No ignition.	Jan. 22	[a] 680	4.06	Do.
Do	246	.....	Do.	Do	[a] 680	4.02	Do.
Do	246	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 12	16.60	44.5	2,681
Do	16.50	44.0	2,666
Do	16.25	45.0	2,769

Average rate of detonation, 2,705 meters (8,870 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 8	14.00	17.68	5.00	0.250
Do	13.75	17.37	5.50	.275
Do	15.25	19.26	5.00	.250

Average height of flame, 18.10 inches.
Average duration of flame, 0.258 millisecond.

Impact Test.

Date, March 4, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	16	4	No explosion.
19	1	Do.	16	1	Explosion.
18	2	No explosion.	15	1	No explosion.
18	1	Explosion.	15	1	Explosion.
17	4	No explosion.	14	5	No explosion.
17	1	Explosion.			

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 158 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 12.....	3	Exploded.	Apr. 12.....	9	Exploded.
Do.....	4	Do.	Do.....	12	Did not explode.
Do.....	5	Do.	Do.....	10	Do.
Do.....	6	Do.	Do.....	10	Do.
Do.....	7	Do.	Do.....	10	Do.

The minimum distance at which no explosion occurs established at 10 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 11.....	219	0.98	31.25	97.66	A	98.70
Do.....	219	.98	31.75	99.22	A	
Do.....	219	.98	31.75	99.22	A	
Do.....	219	.98	28.75	89.84	B	90.36
Do.....	219	.98	29.00	90.62	B	
Do.....	219	.98	29.00	90.62	B	
Do.....	219	.98	28.00	87.50	C	89.06
Do.....	219	.98	28.50	89.06	C	
Do.....	219	.98	29.00	90.62	C	

[a] Including 19 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=108.13$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.98$ $W=219$ grams.

$M=\frac{VPS}{W}=7{,}258$ kilograms per square centimeter (103,230 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 19 Grams of the Original Wrapper.

[Analyst A. L. Hyde.]

Date, February 1, 1912.

	Grams.
Solid	10.5
Liquid (water)	77.0
Gaseous	120.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 7	81.95	0.700	589.9
Do	81.95	.719	606.1
Do	81.95	.690	581.4

[a] Including 9.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 592.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 9	63.25	50.50	12.75
Do	63.25	51.00	12.25
Do	63.50	51.00	12.50

Average compression, 12.5 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 11	63	267	204	15
Do	63	270	207	15
Do	63	265	202	15

Average expansion of bore hole, 204 cubic centimeters (12.44 cubic inches).

COALITE X.

Explosive, Coalite X.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 174 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.92.
Color of explosive, ecru.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, January 10, 1912.

Unit swing on this date, 3.40 inches.

Weight of charge, in grams, 225, 225, 225.

Swing, in inches, 3.43, 3.42, 3.33.

Average swing, in inches, 3.39.

3.39 : 3.40 : : 225 : (226).

Therefore the unit deflective charge of Coalite X is 226 grams.

GAS AND DUST GALLERY No. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Jan. 11	226	8.42	No ignition.	Jan. 22	226	.....	No ignition.
Do	226	8.21	Do.	Do	226	.....	Do.
Jan. 12	226	7.93	Do.	Do	226	.....	Do.
Do	226	7.87	Do.	Do	226	.....	Do.
Do	226	8.20	Do.	Jan. 23	226	.....	Do.
Do	226	7.80	Do.	Do	226	.....	Do.
Do	226	7.79	Do.	Do	226	.....	Do.
Do	226	7.76	Do.	TEST 4.			
Do	226	7.91	Do.				
Do	226	7.89	Do.	Jan. 19	[a] 680	4.26	No ignition.
TEST 3.				Do	[a] 680	4.05	Do.
				Do	[a] 680	3.92	Do.
Jan. 22	226	.....	No ignition.	Do	[a] 680	3.92	Do.
Do	226	.....	Do.	Do	[a] 680	4.13	Do.
Do	226	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 12	16.55	44.5	2,689
Do	15.90	44.5	2,799
Do	16.20	44.5	2,748

Average rate of detonation, 2,745 meters (9,000 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 8	14.50	18.32	7.75	0.388
Do	15.00	18.95	5.25	.262
Do	14.00	17.68	7.00	.350

Average height of flame, 18.32 inches.

Average duration of flame, 0.333 millisecond.

IMPACT TEST.

Date, March 2, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	3	No explosion.	25	1	Explosion.
18	2	Do.	20	1	Do.
20	2	Do.	18	1	Do.
25	1	Do.	16	1	No explosion.
30	1	Explosion.	17	5	Do.

The maximum height at which no explosion occurs established at 17 centimeters (6.69 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 146 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 13	6	Exploded.	Apr. 13	7	Did not explode.
Do	8	Did not explode.	Do	7	Do.
Do	7	Do.			

The minimum distance at which no explosion occurs established at 7 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 10	219	0.92	30.00	93.75	A	94.01
Do	219	.92	30.00	93.75	A	
Do	219	.92	30.25	94.53	A	
Do	219	.92	27.00	84.38	B	85.42
Do	219	.92	27.50	85.94	B	
Do	219	.92	27.50	85.94	B	
Do	219	.92	26.00	81.25	C	81.51
Do	219	.92	26.00	81.25	C	
Apr. 11	219	.92	26.25	82.03	C	

[a] Including 19 grams of the original wrapper.

P=1.911A+0.5B−1.411C=107.35 kilograms per square centimeter.

V=15,000 cubic centimeters. S=0.92. W=219.0 grams.

$M=\frac{VPS}{W}$=6.765 kilograms per square centimeter (96,220 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 31, 1912.	Grams.
Solid	6.5
Liquid (water)	71.0
Gaseous	132.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 6	81.95	1.126	953.6
Do	81.95	1.170	991.2
Do	81.95	1.126	953.6

[a] Including 9.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 966.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 9	63.25	49.25	14.00
Do	63.25	49.50	13.75
Do	63.25	49.25	14.00

Average compression, 13.9 millimeters (0.55 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 10	63	326	263	15
Do	63	319	256	15
Do	63	318	255	15

Average expansion of bore hole, 258 cubic centimeters (15.74 cubic inches).

COALITE NO. 2-D. L.

Explosive, Coalite No. 2-D. L.
Class 4, nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 154 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.89.
Color of explosive, corn.
Consistency, fibrous, fine, dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, October 28, 1910.

Unit swing on this date, 3.15 inches.

Weight of charge, in grams, 287, 287, 287.

Swing, in inches, 3.40, 3.33, 3.30.

Average swing, in inches, 3.34.

3.34 : 3.15 : : 2.87 : (271).

Therefore the unit deflective charge of Coalite No. 2–D. L. is 271 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Oct. 31	271	7.75	No ignition.	Nov. 1	271	.....	No ignition.
Do	271	7.90	Do.	Do	271	.....	Do.
Do	271	8.28	Do.	Do	271	.....	Do.
Do	271	8.04	Do.	Do	271	.....	Do.
Do	271	7.84	Do.	Do	271	.....	Do.
Do	271	7.97	Do.	Do	271	.....	Do.
Nov. 1	271	8.06	Do.	Do	271	.....	Do.
Do	271	7.85	Do.				
Do	271	7.80	Do.	TEST 4.			
Do	271	7.87	Do.				
				Nov. 2	[a]680	3.97	No ignition.
TEST 3.				Do	[a]680	3.99	Do.
				Do	[a]680	4.05	Do.
Nov. 1	271	.....	No ignition.	Do	[a]680	4.07	Do.
Do	271	.....	Do.	Do	[a]680	4.20	Do.
Do	271	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 18	19.95	44.0	2,206
Do	19.70	43.5	2,208
Do	19.15	44.0	2,298

Average rate of detonation, 2,237 meters (7,340 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
June 7	9.25	11.68	4.50	0.225
Do	8.25	10.42	4.25	.213
Do	11.50	14.53	6.00	.300

Average height of flame, 12.21 inches.

Average duration of flame, 0.246 millisecond.

IMPACT TEST.

Date, June 7, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	1	Explosion.
10	1	No explosion.	14	1	Do.
16	1	Do.	13	2	No explosion.
18	1	Explosion.	13	1	Explosion.
17	1	Do.	12	1	Do.
16	1	Do.	11	5	No explosion.

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 143 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 29	8	Did not explode.	May 29	3	Exploded.
Do	6	Do.	June 1	4	Did not explode.
Do	2	Exploded.	Do	4	Do.
Do	4	Did not explode.			

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
May 29	220	0.89	21.75	67.97	A	
Do	220	.89	20.75	64.84	A	66.67
Do	220	.89	21.50	67.19	A	
Do	220	.89	[b] 24.50	61.25	B	
Do	220	.89	[b] 25.25	63.13	B	62.09
Do	220	.89	[b] 24.75	61.88	B	
May 31	220	.89	[b] 24.50	61.25	C	
Do	220	.89	[b] 23.75	59.38	C	60.63
Do	220	.89	[b] 24.50	61.25	C	

[a] Including 20 grams of the original wrapper.
[b] Indicator spring, 0.4 millimeter = 1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=72.90$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.89$. $W=220$ grams.

$M=\frac{VPS}{W}=4{,}424$ kilograms per square centimeter (62,930 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 20 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 5, 1911.

	Grams.
Solid	80.5
Liquid (water)	9.7
Gaseous	124.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
May 31	80.95	0.701	581.6
Do	80.95	.698	579.1
Do	80.95	.701	581.6

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 580.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
June 15	63.25	56.75	6.50
Do	63.00	56.75	6.25
Do	63.25	56.75	6.50

Average compression, 6.4 millimeters (0.25 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 25	63	224	161	15
Do	63	220	157	15
Do	63	220	157	15

Average expansion of bore hole, 158 cubic centimeters (9.64 cubic inches).

COALITE NO. 2-M, L. F.

Explosive, Coalite No. 2-M, L. F.
Class 4, nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 174.5 grams.
Cartridge had not been redipped.
Apparent specific gravity by sand, 1.01.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, January 10, 1912.
Unit swing on this date, 3.40 inches.
Weight of charge, in grams, 310, 310, 310.
Swing, in inches, 3.42, 3.55, 3.53.
Average swing, in inches, 3.50.

3.50 : 3.40 : : 310 : (301).

Therefore the unit deflective charge of Coalite No. 2-M, L. F., is 301 grams.

GAS AND DUST GALLERY No. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 16	301	8.31	No ignition.	Jan. 24	301	.....	No ignition.
Do	301	8.50	Do.	Do	301	.....	Do.
Do	301	8.33	Do.	Do	301	.....	Do.
Jan. 17	301	8.38	Do.	Do	301	.....	Do.
Do	301	8.20	Do.	Do	301	.....	Do.
Do	301	8.23	Do.	Do	301	.....	Do.
Do	301	8.24	Do.	Do	301	.....	Do.
Do	301	8.12	Do.				
Do	301	8.16	Do.	TEST 4.			
Do	301	8.34	Do.				
				Jan. 18	[a] 680	4.09	No ignition.
TEST 3.				Do	[a] 680	4.14	Do.
				Do	[a] 680	3.99	Do.
Jan. 24	301	.....	No ignition.	Do	[a] 680	4.04	Do.
Do	301	.....	Do.	Do	[a] 680	4.03	Do.
Do	301	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 11	16.05	44.5	2,773
Do	15.70	45.0	2,866
Do	16.10	45.0	2,795

Average rate of detonation, 2,811 meters (9,220 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 6	14.00	17.68	7.25	0.362
Do	15.75	19.89	7.00	.350
Do	14.75	18.63	7.50	.375

Average height of flame, 18.73 inches.
Average duration of flame, 0.362 millisecond.

IMPACT TEST.

Date, February 27, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	19	1	Explosion.
20	1	Do.	18	1	No explosion.
25	1	Explosion.	18	1	Explosion.
23	1	No explosion.	17	3	No explosion.
23	1	Explosion.	17	1	Explosion.
22	1	Do.	16	1	Do.
21	1	Do.	15	2	No explosion.
20	1	Do.	15	1	Explosion.
19	2	No explosion.	14	5	No explosion.

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 162 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 4	3	Did not explode.	Apr. 5	3	Exploded.
Do	2	Do.	Do	4	Do.
Do	1	Exploded.	Do	5	Did not explode.
Do	2	Do.	Do	5	Do.
Apr. 5	3	Did not explode.	Do	5	Do.

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 8	215	1.01	20.25	63.28	A	
Do	215	1.01	21.00	65.62	A	64.32
Do	215	1.01	20.50	64.06	A	
Do	215	1.01	20.00	62.50	B	
Do	215	1.01	19.75	61.72	B	62.24
Do	215	1.01	20.00	62.50	B	
Apr. 9	215	1.01	19.00	59.38	C	
Do	215	1.01	19.50	60.94	C	60.68
Do	215	1.01	19.75	61.72	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=68.42$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.01$. $W=215$ grams.

$M=\frac{VPS}{W}=4{,}821$ kilograms per square centimeter (68,570 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 12, 1912.

	Grams.
Solid	70.1
Liquid (water)	4.6
Gaseous	124.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grains. [a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 2	81.95	0.704	604.3
Do	81.95	.730	626.9
Do	81.95	.711	610.4

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 613.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 9	63.25	50.50	12.75
Do	63.25	50.75	12.50
Apr. 10	63.25	51.00	12.25

Average compression, 12.5 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 10	63	226	163	15
Do	63	219	156	15
Do	63	223	160	15

Average expanison of bore hole, 160 cubic centimeters (9.76 cubic inches).

COALITE NO. 3-X.

Explosive, Coalite No. 3-X.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 187 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.08.
Color of explosive, corn.
Consistency, granular, coarse, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum.

Date, October 28, 1910.
Unit swing on this date, 3.15 inches.
Weight of charge, in grams, 267, 267, 267.
Swing, in inches, 3.07, 3.04, 3.09.
Average swing, in inches, 3.07.

3.07 : 3.15 : : 267 : (274).

Therefore the unit deflective charge of Coalite No. 3–X is 274 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 31	274	8.15	No ignition.	Nov. 2	274	.....	No ignition.
Do	274	7.94	Do.	Do	274	.....	Do.
Nov. 1	274	8.12	Do.	Do	274	.....	Do.
Do	274	8.12	Do.	Do	274	.....	Do.
Do	274	7.88	Do.	Do	274	.....	Do.
Do	274	8.00	Do.	Do	274	.....	Do.
Do	274	8.03	Do.	Do	274	.....	Do.
Do	274	8.08	Do.	TEST 4.			
Do	274	8.02	Do.				
Do	274	7.76	Do.	Nov. 2	[a] 680	3.95	No ignition.
TEST 3.				Do	[a] 680	4.05	Do.
				Do	[a] 680	4.04	Do.
Nov. 1	274	.....	No ignition.	Do	[a] 680	4.15	Do.
Do	274	.....	Do.	Do	[a] 680	3.91	Do.
Do	274	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Nov. 17	19.29	43.5	2,255
Do	19.80	43.0	2,172
Do	19.40	43.5	2,242

Average rate of detonation, 2,223 meters (7,290 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 19	5.25	6.63	1.50	0.075
Do	5.00	6.32	1.25	.062
Do	3.50	4.42	1.00	.050

Average height of flame, 5.79 inches.
Average duration of flame, 0.062 millisecond.

Impact Test.

Date, June 8, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	1	Explosion.
16	1	Do.	14	1	Do.
14	1	No explosion.	13	5	No explosion.

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 174 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
June 1	6	Did not explode.	June 1	2	Exploded.
Do	4	Do.	Do	3	Did not explode.
Do	2	Do.	Do	3	Do.
Do	1	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 3	220	1.10	25.50	79.69	A	
Do	220	1.10	26.00	81.25	A	80.47
Do	220	1.10	25.75	80.47	A	
June 2	220	1.10	24.25	75.78	B	
Do	220	1.10	24.00	75.00	B	76.03
Do	220	1.10	24.75	77.32	B	
May 31	220	1.10	23.00	71.87	C	
Do	220	1.10	23.50	73.44	C	72.92
Do	220	1.10	23.50	73.44	C	

[a] Including 20 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=88.91$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.08$. $W=220$ grams.

$M=\frac{VPS}{W}=6{,}547$ kilograms per square centimeter (93,130 pounds per square inch).

Products of Combustion From 200 Grams of the Explosive and 20 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, June 6, 1911.

	Grams.
Solid	44.2
Liquid (water)	63.0
Gaseous	115.2

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
May 31	80.95	0.902	750.6
June 1	80.95	.910	757.4
Do	80.95	.896	745.6

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 751.2.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
May 22	63.50	51.25	12.25
Do	63.50	51.50	12.00
Do	63.50	51.50	12.00

Average compression, 12.1 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 25	63	268	205	15
Do	63	270	207	15
Do	63	275	212	15

Average expansion of bore hole, 208 cubic centimeters (12.69 cubic inches).

COALITE NO. 3-XA.

Explosive, Coalite No. 3-XA.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8 inches.
Average weight, 128 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, corn.
Consistency, granular with white crystals, very fine, dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, April 10, 1911.

Unit swing on this date, 2.93 inches.

Weight of charge, in grams, 274, 274, 274.

Swing, in inches, 2.78, 2.78, 2.67.

Average swing, in inches, 2.74.

$$2.74 : 2.93 : : 274 : (293).$$

Therefore the unit deflective charge of Coalite No. 3–XA is 293 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Apr. 14	293	8.27	No ignition.	Apr. 19	293	.....	No ignition.
Do	293	8.09	Do.	Do	293	.....	Do.
Do	293	8.07	Do.	Do	293	.....	Do.
Apr. 15	293	8.07	Do.	Apr. 20	293	.....	Do.
Do	293	8.08	Do.	Do	293	.....	Do.
Do	293	8.10	Do.	Do	293	.....	Do.
Do	293	8.00	Do.	Do	293	.....	Do.
Do	293	8.21	Do.				
Do	293	7.94	Do.				
Do	293	7.97	Do.	TEST 4.			
					Grams.	*Per cent.*	
				Apr. 21	[a] 680	3.98	No ignition.
TEST 3.				Do	[a] 680	3.99	Do.
	Grams.	*Per cent.*		Do	[a] 680	3.80	Do.
Apr. 19	293	.....	No ignition.	Apr. 22	[a] 680	3.96	Do.
Do	293	.....	Do.	Do	[a] 680	4.07	Do.
Do	293	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation.
	Millimeters.	*Meters.*	*Meters.*
Apr. 17	14.16	46.00	3,249
Do	14.56	46.25	3,177
Do	14.91	46.25	3,102

Average rate of detonation, 3,176 meters (10,420 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 3	14.50	18.32	8.00	0.400
Do	18.25	23.05	6.25	.312
Do	18.50	23.37	7.25	.362

Average height of flame, 21.58 inches.

Average duration of flame, 0.358 millisecond.

Impact Test.

Date, July 25, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
16	1	Explosion.	15	5	No explosion.
14	1	No explosion.			

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 150 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 18	2	Did not explode.	Jan. 18	2	Did not explode.
Do	1	Exploded.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 21	221	0.93	24.75	77.34	A	
Nov. 22	221	.93	24.50	76.56	A	76.04
Do	221	.93	23.75	74.22	A	
Nov. 14	221	.93	23.25	72.66	B	
Do	221	.93	24.50	76.56	B	74.48
Do	221	.93	23.75	74.22	B	
Nov. 15	221	.93	22.50	70.31	C	
Nov. 16	221	.93	22.50	70.31	C	70.83
Do	221	.93	23.00	71.87	C	

[a] Including 21 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=82.61$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.93$. $W=221$ grams.

$M=\frac{VPS}{W}=5{,}215$ kilograms per square centimeter.

Products of Combustion from 200 Grams of the Explosive and 21 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, July 10, 1911.

	Grams.
Solid	10.1
Liquid (water)	63.5
Gaseous	128.2

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 20	81.95	0.925	774.9
Do	81.95	.884	740.2
Do	81.95	.908	760.5

[a] Including 10.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 758.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Nov. 3	63.50	48.75	14.75
Do	63.50	48.50	15.00
Do	63.50	49.00	14.50

Average compression, 14.8 millimeters (0.58 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*° C.*
Jan. 16	62	274	212	20
Do	62	274	212	17
Do	62	272	210	18

Average expansion of bore hole, 211 cubic centimeters (12.87 cubic inches).

COALITE NO. 3-XB.

Explosive, Coalite No. 3-XB.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 128 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.92.
Color of explosive, drab.
Consistency, granular and fibrous with white crystals, very fine, dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, April 10, 1911.

Unit swing on this date, 2.93 inches.

Weight of charge, in grams, 293, 293, 293.

Swing, in inches, 2.90, 2.95, 2.94.

Average swing, in inches, 2.93.

2.93 : 2.93 : : 293 : (293).

Therefore the unit deflective charge of Coalite No. 3–XB is 293 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Apr. 15	293	8.05	No ignition.	Apr. 20	293	.....	No ignition.
Apr. 17	293	8.11	Do.	Do	293	.....	Do.
Do	293	7.85	Do.	Do	293	.....	Do.
Do	293	8.13	Do.	Do	293	.....	Do.
Do	293	8.26	Do.	Do	293	.....	Do.
Do	293	8.08	Do.	Do	293	.....	Do.
Do	293	8.04	Do.	Do	293	.....	Do.
Do	293	8.16	Do.				
Do	293	7.94	Do.	TEST 4.			
Do	293	8.07	Do.				
				Apr. 22	a 680	4.00	No ignition.
TEST 3.				May 17	a 680	4.06	Do.
				Do	a 680	4.01	Do.
Apr. 20	293	.....	No ignition.	Do	a 680	4.00	Do.
Do	293	.....	Do.	Do	a 680	4.02	Do.
Do	293	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Apr. 17	13.70	46.75	3,412
Do	13.41	46.75	3,486
Apr. 18	12.70	43.50	3,425

Average rate of detonation, 3,441 meters (11,290 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 11	5.75	7.26	1.75	0.875
Do	11.00	13.89	4.50	2.250
Do	5.75	7.26	1.25	.625

Average height of flame, 9.47 inches.

Average duration of flame, 1.25 milliseconds.

Impact Test.

Date, July 25, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
16	1	No explosion.	21	1	No explosion.
18	1	Do.	21	1	Explosion.
22	1	Explosion.	20	5	No explosion.

The maximum height at which no explosion occurs established at 20 centimeters (7.87 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 148 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 18.....	2	Did not explode.	Jan. 18.....	2	Did not explode.
Do.....	1	Exploded.	Do.....	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 9.....	220	0.92	27.00	84.37	A	
Nov. 10.....	220	.92	26.25	82.03	A	83.07
Do.....	220	.92	26.50	82.81	A	
Do.....	220	.92	24.00	75.00	B	
Do.....	220	.92	24.00	75.00	B	73.97
Do.....	220	.92	23.00	71.90	B	
Do.....	220	.92	24.50	76.56	C	
Do.....	220	.92	24.00	75.00	C	75.26
Do.....	220	.92	23.75	74.23	C	

[a] Including 20 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=89.54$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.92$. $W=220$ grams.

$M=\frac{VPS}{W}=5{,}617$ kilograms per square centimeter (79,900 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 20 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, July 11, 1911.	Grams.
Solid	18.7
Liquid (water)	62.5
Gaseous	125.1

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 2	81.95	0.912	767.4
Dec. 14	81.95	.927	780.1
Dec. 27	81.95	.948	798.0

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 781.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Nov. 2	63.50	49.00	14.50
Do	63.50	48.75	14.75
Do	63.50	48.50	15.00

Average compression, 14.8 millimeters (0.58 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 12	63	287	224	15
Do	63	298	235	15
Do	63	294	231	15

Average expansion of bore hole, 230 cubic centimeters (14.03 cubic inches).

COALITE NO. 3-XC.

Explosive, Coalite No. 3-XC.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Potts Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 169 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, ecru.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, January 10, 1912.

Unit swing on this date, 3.40 inches.

Weight of charge, in grams, 255, 255, 255.

Swing, in inches, 3.50, 3.42, 3.48.

Average swing, in inches, 3.47.

3.47 : 3.40 : : 255 : (250).

Therefore the unit deflective charge of Coalite No. 3–XC is 250 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 13	250	8.21	No ignition.	Jan. 23	250	.....	No ignition.
Do	250	8.22	Do.	Do	250	.....	Do.
Jan. 15	250	8.00	Do.	Do	250	.....	Do.
Do	250	8.23	Do.	Do	250	.....	Do.
Do	250	8.50	Do.	Jan. 24	250	.....	Do.
Do	260	8.25	Do.	Do	250	.....	Do.
Do	250	8.14	Do.	Do	250	.....	Do.
Do	260	7.99	Do.				
Do	260	7.92	Do.	TEST 4.			
Do	250	8.11	Do.				
				Jan. 18	[a]680	4.20	No ignition.
TEST 3.				Do	[a]680	4.14	Do.
				Do	[a]680	4.07	Do.
Jan. 23	250	.....	No ignition.	Do	[a]680	4.03	Do.
Do	250	.....	Do.	Do	[a]680	4.00	Do.
Do	250	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 11	17.40	44.5	2,557
Do	17.60	44.0	2,500
Jan. 12	17.65	45.0	2,550

Average rate of detonation, 2,536 meters (8,320 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 8	13.50	17.05	5.00	0.250
Do	13.75	17.37	6.00	.300
Do	13.50	17.05	7.25	.362

Average height of flame, 17.16 inches.

Average duration of flame, 0.304 millisecond.

Impact Test.

Date, February 27, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	13	5	No explosion.
15	1	Explosion.	14	5	Do.

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 150 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 17	6	Did not explode.	Apr. 17	2	Did not explode.
Do	5	Do.	Do	2	Do.
Do	4	Do.	Do	1	Exploded.
Do	3	Do.	Do	2	Did not explode.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 10	221	0.93	27.25	85.16	A	
Do	221	.93	27.25	85.16	A	85.16
Do	221	.93	27.25	85.16	A	
Apr. 9	221	.93	25.50	79.69	B	
Do	221	.93	26.50	82.81	B	80.99
Do	221	.93	25.75	80.47	B	
Do	221	.93	27.00	84.38	C	
Do	221	.93	27.75	86.72	C	85.16
Do	221	.93	27.00	84.38	C	

[a] Including 21 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=83.08$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.93$. $W=221.0$ grams.

$M=\frac{VPS}{W}=5{,}244$ kilograms per square centimeter (74,590 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 21 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, January 31, 1912.

	Grams.
Solid	10.8
Liquid (water)	69.0
Gaseous	129.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 5	81.95	1.296	1,088.9
Do	81.95	1.316	1,105.8
Mar. 6	81.95	1.346	1,131.1

[a] Including 10.5 grams of the original wrapper.

Average large calories per kilogram of the explosive, 1,108.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 9	63.50	51.25	12.25
Do	63.50	51.00	12.50
Do	63.25	50.75	12.50

Average compression, 12.4 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 10	63	317	254	15
Do	63	310	247	15
Do	63	309	246	15

Average expansion of bore hole, 249 cubic centimeters (15.19 cubic inches).

COAL SPECIAL NO. 2-W.

Explosive, Coal special No. 2-W.
Class 4, nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8½ inches.
Average weight, 179 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.08.
Color of explosive, corn.
Consistency, fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, October 13, 1910.
Unit swing on this date, 3.28 inches.
Weight of charge, in grams, 290, 290, 290.
Swing, in inches, 3.35, 3.45, 3.52.
Average swing, in inches, 3.44.

3.44 : 3.28 : : 290 : (277).

Therefore the unit deflective charge of Coal special No. 2–W is 277 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 13	277	7.83	No ignition.	Oct. 18	277	.....	No ignition.
Do	277	7.80	Do.	Do	277	.....	Do.
Do	277	7.81	Do.	Do	277	.....	Do.
Oct. 14	277	7.95	Do.	Do	277	.....	Do.
Do	277	7.94	Do.	Do	277	.....	Do.
Do	277	8.04	Do.	Do	277	.....	Do.
Do	277	7.80	Do.	Do	277	.....	Do.
Do	277	7.92	Do.	TEST 4.			
Do	277	7.92	Do.				
Do	277	7.92	Do.	Oct. 20	[a]680	3.96	No ignition.
TEST 3.				Do	[a]680	3.95	Do.
				Do	[a]680	4.15	Do.
Oct. 18	277	.....	No ignition.	Do	[a]680	4.04	Do.
Do	277	.....	Do.	Do	[a]680	4.13	Do.
Do	277	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 20	12.40	43.5	3,508
Oct. 26	12.30	43.5	3,537
Do	12.23	43.5	3,557

Average rate of detonation, 3,534 meters (11,590 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
June 6	14.25	18.00	5.75	0.288
Do	16.00	20.21	6.00	.300
Do	13.00	16.42	5.75	.288

Average height of flame, 18.21 inches.
Average duration of flame, 0.292 millisecond.

IMPACT TEST.

Date, May 19, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	10	1	No explosion.
12	1	Do.	11	5	Do.
8	1	No explosion.			

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 174 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 13	8	Did not explode.	May 15	7	Exploded.
Do	6	Do.	Do	8	Did not explode.
Do	4	Exploded.	Do	8	Exploded.
May 15	5	Do.	Do	9	Did not explode.
Do	6	Do.	Do	9	Do.
Do	7	Did not explode.	Do	9	Do.

The minimum distance at which no explosion occurs established at 9 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
May 10	214	1.08	25.50	63.75	A	
Do	214	1.08	25.75	64.38	A	64.17
Do	214	1.08	25.75	64.38	A	
May 11	214	1.08	23.50	58.75	B	
Do	214	1.08	24.00	60.00	B	59.38
Do	214	1.08	23.75	59.38	B	
Do	214	1.08	24.00	60.00	C	
Do	214	1.08	23.75	59.38	C	60.00
Do	214	1.08	24.25	60.62	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=67.66$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.08$. $W=214$ grams.

$M=\frac{VPS}{W}=5{,}122$ kilograms per square centimeter (72,860 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, May 24, 1911.	Grams.
Solid	69.0
Liquid (water)	9.4
Gaseous	121.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
May 24	80.95	0.811	693.1
May 25	80.95	.806	688.8
Do	80.95	.807	689.6

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 690.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
May 20	63.50	47.75	15.75
Do	63.50	47.75	15.75
Do	63.50	47.75	15.75

Average compression, 15.8 millimeters (0.62 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 23	63	220	157	15
Do	63	218	155	15
Do	63	220	157	15

Average expansion of bore hole, 156 cubic centimeters (9.52 cubic inches).

COAL SPECIAL NO. 3–C.

Explosive, Coal special No. 3–C.
Class 4, Nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 185 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.12.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, moderately cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, November 3, 1909.

Unit swing on this date, 3.06 inches.

Weight of charge, in grams, 300, 300, 300.

Swing, in inches, 3.19, 3.17, 3.22.

Average swing, in inches, 3.19.

3.19 : 3.06 : : 300 : (288).

Therefore the unit deflective charge of Coal special No. 3–C is 288 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Nov. 5	288	8.26	No ignition.	Nov. 8	288	.....	No ignition.
Do	288	8.26	Do.	Do	288	.....	Do.
Do	288	8.26	Do.	Do	288	.....	Do.
Do	288	8.43	Do.	Do	288	.....	Do.
Do	288	8.43	Do.	Do	288	.....	Do.
Do	288	8.26	Do.	Do	288	.....	Do.
Do	288	8.26	Do.	Do	288	.....	Do.
Nov. 6	288	8.26	Do.				
Do	288	8.43	Do.				
Do	288	8.43	Do.	TEST 4.			
				Nov. 8	[a] 680	3.77	No ignition.
TEST 3.				Do	[a] 680	3.95	Do.
				Nov. 9	[a] 680	3.95	Do.
Nov. 8	288	.....	No ignition.	Do	[a] 680	4.07	Do.
Do	288	.....	Do.	Do	[a] 680	4.13	Do.
Do	288	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Nov. 5	14.11	43.0	3,047
Do	14.72	43.0	2,921
Nov. 6	14.51	43.0	2,963

Average rate of detonation, 2,977 meters (9,760 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 21	12.25	15.47	4.75	0.238
Do	12.50	15.79	4.75	.238
Do	11.25	14.21	4.50	.225

Average height of flame, 15.16 inches.

Average duration of flame, 0.234 millisecond.

IMPACT TEST.

Date, June 24, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	24	1	Explosion.
22	1	Do.	22	1	No explosion.
24	1	Do.	22	1	Explosion.
26	1	Explosion.	21	2	No explosion.
25	1	No explosion.	21	1	Explosion.
25	1	Explosion.	20	4	No explosion.

The maximum height at which no explosion occurs established at 20 centimeters (7.87 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 180 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
July 16	10	Did not explode.	July 16	10	Did not explode.
Do	9	Exploded.	July 18	10	Do.

The minimum distance at which no explosion occurs established at 10 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 16	213	1.12	27.00	67.50	A	
Do	213	1.12	26.75	66.88	A	67.29
Do	213	1.12	27.00	67.50	A	
Sept. 14	213	1.12	26.25	65.62	B	
Do	213	1.12	25.50	63.75	B	64.37
Sept. 15	213	1.12	25.50	63.75	B	
Do	213	1.12	24.50	61.25	C	
Do	213	1.12	25.00	62.50	C	62.29
Sept. 24	213	1.12	25.25	63.12	C	

[a] Including 13 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=72.88$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.12$. $W=213$ grams.

$M=\frac{VPS}{W}=5{,}748$ kilograms per square centimeter (81,760 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 22, 1910.

	Grams.
Solid	79.8
Liquid (water)	13.2
Gaseous	118.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1910).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
June 13	81.95	0.773	670.6
June 14	81.95	.796	690.8
Do	81.95	.775	672.3

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 677.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 12	68.00	51.00	12.00
Do	63.00	51.00	12.00
Do	63.00	50.75	12.25

Average compression, 12.1 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
June 2	63	226	163	15
Do	63	213	150	15
Do	63	223	160	15

Average expansion of bore hole, 158 cubic centimeters (9.64 cubic inches).

COLLIER POWDER B, N. F.

Explosive, Collier powder B, N. F.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 168 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.99.
Color of explosive, drab.
Consistency, granular and fibrous, fine, dry, soft, moderately cohesive.

Unit deflective charge as determined by ballistic pendulum

Date, January 2, 1913.

Unit swing on this date, 3.40 inches.

Weight of charge, in grams, 230, 230, 230.

Swing, in inches, 3.51, 3.43, 3.52.

Average swing, in inches, 3.49.

3.49 : 3.40 : : 230 : (224).

Therefore the unit deflective charge of Collier powder B, N. F., is 224 grams.

GAS AND DUST GALLERY No. 1.

Date (1913).	Weight of charge.	Methane and ethane.	Result.	Date (1913).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Jan. 6	224	7.88	No ignition.	Jan. 10	224	.....	No ignition.
Jan. 7	224	7.94	Do.	Do	224	.....	Do.
Do	224	8.13	Do.	Jan. 11	224	.....	Do.
Do	224	7.90	Do.	Do	224	.....	Do.
Do	224	8.03	Do.	Do	224	.....	Do.
Do	224	8.24	Do.	Do	224	.....	Do.
Do	224	8.16	Do.	Do	224	.....	Do.
Do	224	7.99	Do.				
Do	224	7.81	Do.	TEST 4.			
Do	224	7.76	Do.				
				Jan. 8	a 680	3.93	No ignition.
TEST 3.				Do	a 680	4.11	Do.
				Do	a 680	4.00	Do.
Jan. 10	224	.....	No ignition.	Do	a 680	3.99	Do.
Do	224	.....	Do.	Do	a 680	3.94	Do.
Do	224	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1913).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 7	12.60	42.0	3,333
Jan. 8	12.40	42.0	3,387
Jan. 14	12.75	42.0	3,294

Average rate of detonation, 3,338 meters (10,950 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	10.50	13.26	6.25	0.312
Do	9.75	12.32	6.50	.325
Do	10.50	13.26	6.75	.338

Average height of flame, 12.95 inches.

Average duration of flame, 0.325 millisecond.

IMPACT TEST.

Date, March 4, 1913.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	No explosion.	35	3	No explosion.
38	1	Explosion.	35	1	Explosion.
34	1	No explosion.	34	1	Do.
36	1	Do.	33	2	No explosion.
37	3	Do.	33	1	Explosion.
37	1	Explosion.	32	3	No explosion.
36	1	No explosion.	32	1	Explosion.
36	1	Explosion.	31	5	No explosion.

The maximum height at which no explosion occurs established at 31 centimeters (12.20 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 159 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 26	4	Did not explode.	Feb. 26	4	Did not explode.
Do	3	Exploded.	Do	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 14	217	0.99	23.25	96.88	A	96.18
Feb. 15	217	.99	22.75	94.79	A	
Do	217	.99	23.25	96.88	A	
Do	217	.99	22.75	94.79	B	
Feb. 18	217	.99	22.75	94.79	B	94.79
Do	217	.99	22.75	94.79	B	
Do	217	.99	22.00	91.67	C	
Do	217	.99	22.50	93.75	C	92.71
Do	217	.99	22.25	92.71	C	

[a] Including 17 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=100.38$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.99$. $W=217$ grams.

$M=\frac{VPS}{W}=6{,}869$ kilograms per square centimeter (97,700 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 16, 1913.	Grams.
Solid	21.2
Liquid (water)	59.5
Gaseous	129.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 14	81.95	1.135	970.1
Feb. 15	81.95	1.139	973.6
Do	81.95	1.119	956.4

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 966.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 25	63.50	49.00	14.50
Do	63.50	49.50	14.00
Mar. 18	63.50	49.50	14.00

Average compression, 14.2 millimeters (0.56 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1913).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 5	63	315	252	15
Do	63	310	247	15
Do	63	305	242	15

Average expansion of bore hole, 247 cubic centimeters (15.07 cubic inches).

COLLIER POWDER KN.

Explosive, Collier powder KN.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8 inches.
Average weight, 137 grams.
Two inches of one end of cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.99.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, June 3, 1912.

Unit swing on this date, 3.27 inches.

Weight of charge, in grams, 225, 225, 225.

Swing, in inches, 3.32, 3.36, 3.26.

Average swing, in inches, 3.31.

3.31 : 3.27 : : 225 : (222).

Therefore the unit deflective charge of Collier powder KN is 222 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
June 4	222	8.21	No ignition.	June 6	222	.....	No ignition.
Do	222	7.81	Do.	Do	222	.....	Do.
Do	222	8.12	Do.	Do	222	.....	Do.
Do	222	8.08	Do.	Do	222	.....	Do.
Do	222	7.94	Do.	Do	222	.....	Do.
Do	222	8.37	Do.	Do	222	.....	Do.
June 5	222	8.00	Do.	Do	222	.....	Do.
Do	222	7.79	Do.				
Do	222	8.32	Do.	TEST 4.			
Do	222	8.27	Do.				
				June 5	[a] 680	4.17	No ignition.
TEST 3.				June 6	[a] 680	4.07	Do.
				Do	[a] 680	4.18	Do.
June 6	222	.....	No ignition.	Do	[a] 680	4.18	Do.
Do	222	.....	Do.	Do	[a] 680	4.07	Do.
Do	222	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 7	12.35	44.0	3,563
Do	12.75	44.5	3,490
Do	12.35	44.0	3,563

Average rate of detonation, 3,539 meters (11,610 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 5	4.50	5.68	1.25	0.062
Sept. 13	6.50	8.21	1.50	.075
Do	8.50	10.74	1.50	.075

Average height of flame, 8.21 inches.

Average duration of flame, 0.071 millisecond.

IMPACT TEST.

Date, August 20, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	26	1	Explosion.
20	1	No explosion.	25	1	Do.
30	1	Explosion.	24	5	No explosion.
27	1	Do.			

The maximum height at which no explosion occurs established at 24 centimeters (9.45 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 159 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 9.....	3	Exploded.	Sept. 9.	4	Did not explode.
Do.....	4	Did not explode.	Do.....	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 26.....	216	0.99	32.75	102.34	A	
Do.....	216	.99	33.75	105.47	A	104.17
Aug. 27.....	216	.99	33.50	104.69	A	
Do.....	216	.99	30.00	93.75	B	
Do.....	216	.99	30.25	94.53	B	93.75
Do.....	216	.99	29.75	92.97	B	
Do.....	216	.99	29.25	91.41	C	
Do.....	216	.99	29.50	92.19	C	92.97
Do.....	216	.99	30.50	95.31	C	

[a] Including 16 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=114.76$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.99$. $W=216$ grams.

$M=\frac{VPS}{W}=7{,}890$ kilograms per square centimeter (112,220 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 16 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 13, 1912.	Grams.
Solid	4.0
Liquid (water)	71.5
Gaseous	126.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Sept. 23	81.95	1.250	1,074.2
Do	81.95	1.279	1,099.4
Sept. 25	81.95	1.229	1,056.0

[a] Including 8.0 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,076.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 23	63.25	45.00	18.25
Do	63.25	45.00	18.25
Sept. 24	63.25	44.75	18.50

Average compression, 18.3 millimeters (0.72 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Oct. 17	62	336	274	15
Do	62	326	264	15
Dec. 17	63	328	265	15

Average expansion of bore hole, 268 cubic centimeters (16.35 cubic inches).

COLLIER POWDER NO. X.

Explosive, Collier powder No. X.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 148 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.06.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, August 15, 1910.
Unit swing on this date, 3.18 inches.
Weight of charge, in grams, 265, 265, 265.
Swing, in inches, 3.04, 3.05, 3.18.
Average swing, in inches, 3.09.

3.09 : 3.18 : : 265 : (273).

Therefore the unit deflective charge of Collier powder No. X is 273 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Aug. 18	273	7.90	No ignition.	Aug. 19	273	.....	No ignition.
Do	273	8.06	Do.	Do	273	.....	Do.
Do	273	8.18	Do.	Do	273	.....	Do.
Do	273	7.90	Do.	Do	273	.....	Do.
Do	273	8.04	Do.	Do	273	.....	Do.
Do	273	7.90	Do.	Do	273	.....	Do.
Do	273	8.16	Do.	Do	273	.....	Do.
Aug. 24	273	8.05	Do.				
Do	273	7.90	Do.	TEST 4.			
Do	273	8.25	Do.				
				Aug. 22	[a] 680	4.10	No ignition.
TEST 3.				Aug. 23	[a] 680	4.01	Do.
				Do	[a] 680	4.10	Do.
Aug. 19	273	.....	No ignition.	Do	[a] 680	4.10	Do.
Do	273	.....	Do.	Aug. 27	[a] 680	4.05	Do.
Do	273	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 24	17.78	43.0	2,418
Aug. 30	18.78	43.5	2,316
Aug. 31	17.99	43.5	2,418

Average rate of detonation, 2,384 meters (7,820 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 13	9.00	11.37	4.60	0.200
Do	13.25	16.74	5.00	.250
Do	11.00	13.90	4.25	.213

Average height of flame, 14 inches.
Average duration of flame, 0.221 millisecond.

IMPACT TEST.

Date, February 1, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
18	1	Explosion.	16	1	No explosion.
17	3	No explosion.	16	1	Explosion.
17	1	Explosion.	15	5	No explosion.

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 171 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 11	3	Did not explode.	Feb. 11	2	Did not explode.
Do	2	Do.	Do	2	Do.
Do	1	Exploded.			

The minimum distance at which no explosion occurs established at 2 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 4	218	1.06	30.20	94.40	A	94.90
Do	218	1.06	30.50	95.30	A	
Feb. 6	218	1.06	30.40	95.00	A	
Do	218	1.06	28.75	89.84	B	89.06
Feb. 7	218	1.06	28.00	87.50	B	
Do	218	1.06	28.75	89.84	B	
Feb. 6	218	1.06	26.75	83.59	C	83.85
Do	218	1.06	27.25	85.15	C	
Feb. 7	218	1.06	26.50	82.81	C	

[a] Including 18 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=107.57$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.06$. $W=218$ grams.

$M=\frac{VPS}{W}=7{,}846$ kilograms per square centimeter (111,600 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, September 12, 1910.	Grams.
Solid	17.0
Liquid (water)	77.0
Gaseous	103.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 8	81.55	1.170	991.4
Do	81.55	1.137	963.3
Do	81.55	1.163	985.2

[a] Including 9 grams of the original wrapper.

Average large calories per kilogram of explosive, 980.0.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 24	63.75	52.25	11.50
Do	63.50	52.00	11.50
Do	63.50	52.00	11.50

Average compression, 11.5 millimeters (0.45 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Jan. 27	63	212	149	15
Do	63	209	146	15
Do	63	212	149	15

Average expansion of bore hole, 148 cubic centimeters (9.03 cubic inches).

COLLIER POWDER X, L. F.

Explosive, Collier powder X, L. F.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:

Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 173 grams.
Only one end of cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.06.
Color of explosive, corn.
Consistency, powdered and granular, very fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, December 28, 1911.
Unit swing on this date, 3.42 inches.
Weight of charge, in grams, 250, 250, 250.
Swing, in inches, 3.45, 3.43, 3.30.
Average swing, in inches, 3.39.

3.39 : 3.42 : : 250 : (252).

Therefore the unit deflective charge of Collier powder X, L. F., is 252 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911–12).	Weight of charge.	Methane and ethane.	Result.	Date (1911–12).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Dec. 30	252	8.21	No ignition.	Jan. 8	252	.....	No ignition.
Do	252	8.07	Do.	Do	252	.....	Do.
Do	252	8.37	Do.	Do	252	.....	Do.
Do	252	7.95	Do.	Do	252	.....	Do.
Do	252	7.95	Do.	Do	252	.....	Do.
Jan. 2	252	8.09	Do.	Do	252	.....	Do.
Do	252	8.42	Do.	Do	252	.....	Do.
Do	252	8.04	Do.	TEST 4.			
Do	252	8.29	Do.				
Do	252	8.16	Do.	Jan. 5	[a] 680	4.09	No ignition.
TEST 2.				Do	[a] 680	3.85	Do.
				Jan. 6	[a] 680	4.14	Do.
Jan. 8	252	.....	No ignition.	Do	[a] 680	4.01	Do.
Do	252	.....	Do.	Do	[a] 680	4.10	Do.
Do	252	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 29	14.50	44.5	3,068
Do	14.65	44.5	3,038
Do	14.50	44.0	3,034

Average rate of detonation, 3,047 meters (9,990 feet per second).

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 8	9.00	11.37	3.00	0.150
Do	14.50	18.32	4.50	.225
Do	14.00	17.68	4.00	.200

Average height of flame, 15.79 inches.
Average duration of flame, 0.192 millisecond.

Impact Test.

Date, February 26, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	16	3	No explosion.
18	1	Do.	16	1	Explosion.
20	1	Explosion.	15	1	No explosion.
19	3	No explosion.	15	1	Explosion.
19	1	Explosion.	14	1	No explosion.
18	1	No explosion.	14	1	Explosion.
18	1	Explosion.	13	5	No explosion.
17	1	Do.			

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 171 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 4	3	Exploded.	Apr. 4	6	Did not explode.
Do	4	Do.	Do	6	Do.
Do	5	Do.	Do	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 5	213	1.06	31.25	97.66	A	
Apr. 6	213	1.06	31.50	98.44	A	98.18
Do	213	1.06	31.50	98.44	A	
Apr. 5	213	1.06	28.50	89.06	B	
Do	213	1.06	28.00	87.50	B	88.80
Do	213	1.06	28.75	89.84	B	
Apr. 4	213	1.06	28.25	88.28	C	
Apr. 5	213	1.06	28.75	89.84	C	88.80
Do	213	1.06	28.25	88.28	C	

[a] Including 13 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=106.73$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.06$. $W=213.0$ grams.

$M=\frac{VPS}{W}=7{,}967$ kilograms per square centimeter (113,320 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 13 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, January 24, 1912.

	Grams.
Solid	19.7
Liquid (water)	70.0
Gaseous	108.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 1	81.95	1.229	1,070.9
Mar. 2	81.95	1.225	1,093.7
Mar. 4	81.95	1.249	1,088.4

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,084.3.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 28	63.50	48.75	14.75
Do	63.50	49.25	14.25
Do	63.50	49.00	14.50

Average compression, 14.5 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 23	62	240	178	15
Do	62	238	176	15
Do	62	246	184	15

Average expansion of bore hole, 179 cubic centimeters (10.92 cubic inches).

COLLIER POWDER NO. 5–L. F.

Explosive, Collier powder No. 5–L. F.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 151 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.06.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, August 15, 1910.

Unit swing on this date, 3.18 inches.

Weight of charge, in grams, 236, 236, 236.

Swing, in inches, 3.26, 3.22, 3.23.

Average swing, in inches, 3.24.

3.24 : 3.18 : : 236 : (232).

Therefore the unit deflective charge of Collier powder No. 5–L. F. is 232 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 17	232	7.90	No ignition.	Aug. 19	232	.....	No ignition.
Do	232	7.90	Do.	Do	232	.....	Do.
Do	232	7.84	Do	Do	232	.....	Do.
Aug. 18	232	8.14	Do.	Do	232	.....	Do.
Do	232	7.85	Do.	Do	232	.....	Do.
Do	232	8.37	Do.	Do	232	.....	Do.
Do	232	8.10	Do.	Do	232	.....	Do.
Do	232	8.02	Do.				
Do	232	7.99	Do.	TEST 4.			
Do	232	9.34	Do.				
				Aug. 22	[a] 680	4.22	No ignition.
TEST 3.				Do	[a] 680	4.20	Do.
				Do	[a] 680	4.15	Do.
Aug. 19	232	.....	No ignition.	Do	[a] 680	4.10	Do.
Do	232	.....	Do.	Do	[a] 680	4.20	Do.
Do	232	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 29	15.07	43.5	2,887
Aug. 30	15.57	43.5	2,794
Do	15.04	43.5	2,892

Average rate of detonation, 2,858 meters (9,370 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 13	16.50	20.84	7.25	0.363
Do	18.25	23.05	8.00	.400
Do	14.00	17.68	6.25	.313

Average height of flame, 20.52 inches.

Average duration of flame, 0.359 millisecond.

IMPACT TEST.

Date, January 31, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
26	1	Explosion.	17	1	No explosion.
24	1	Do.	17	1	Explosion.
22	1	Do.	16	1	No explosion.
18	1	Do.	16	1	Explosion.
16	1	No explosion.	15	5	No explosion.

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 171 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 30	4	Did not explode.	Jan. 30	3	Exploded.
Do	3	Do.	Do	4	Did not explode.
Do	2	Exploded.	Do	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 3	217	1.06	32.40	101.25	A	
Feb. 4	217	1.06	33.40	104.38	A	103.54
Do	217	1.06	33.60	105.00	A	
Jan. 24	217	1.06	31.00	96.88	B	
Feb. 6	217	1.06	31.50	98.44	B	97.40
Do	217	1.06	31.00	96.88	B	
Jan. 24	217	1.06	29.80	93.12	C	
Do	217	1.06	29.60	92.50	C	93.12
Do	217	1.06	30.00	93.75	C	

[a] Including 17 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=115.17$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.06$. $W=217$ grams.

$M=\frac{VPS}{W}=8{,}439$ kilograms per square centimeter (120,040 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, September 12, 1910.	Grams.
Solid	15.0
Liquid (water)	80.0
Gaseous	110.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 7	81.55	1.297	1,105.0
Do	81.55	1.295	1,103.2
Do	81.55	1.294	1,102.4

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,103.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan 24	63.25	48.50	14.75
Do	63.50	48.75	14.75
Do	63.50	49.00	14.50

Average compression, 14.7 millimeters (0.58 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Jan. 27	63	254	191	15
Do	63	249	186	15
Do	63	250	187	15

Average expansion of bore hole, 188 cubic centimeters (11.47 cubic inches).

COLLIER POWDER NO. 5 SPECIAL.

Explosive, Collier powder No. 5 special.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8 inches.
Average weight, 138 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, August 15, 1910.

Unit swing on this date, 3.18 inches.

Weight of charge, in grams, 230, 230, 230.

Swing, in inches, 3.12, 3.07, 3.12.

Average swing, in inches, 3.10.

3.10 : 3.18 : : 230 : (236).

Therefore the unit deflective charge of Collier powder No. 5 special is 236 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Aug. 17	236	8.64	No ignition.	Aug. 18	236	.....	No ignition.
Do	236	8.77	Do.	Do	236	.....	Do.
Do	236	8.78	Do.	Aug. 19	236	.....	Do.
Do	236	8.55	Do.	Do	236	.....	Do.
Do	236	8.37	Do.	Do	236	.....	Do.
Do	236	8.14	Do.	Do	236	.....	Do.
Do	236	8.02	Do.	Do	236	.....	Do.
Do	236	7.86	Do.				
Do	236	7.90	Do.				
Do	236	8.20	Do.				
				TEST 4.			
				Aug. 20	[a] 680	3.87	No ignition.
TEST 3.				Do	[a] 680	4.06	Do.
				Aug. 22	[a] 680	3.95	Do.
Aug. 18	236	.....	No ignition.	Do	[a] 680	3.95	Do.
Do	236	.....	Do.	Do	[a] 680	4.20	Do.
Do	236	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 27	17.22	43.5	2,526
Do	17.07	43.5	2,548
Aug. 29	17.06	43.5	2,550

Average rate of detonation, 2,541 meters (8,330 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 13	16.00	20.21	7.50	0.375
Do	15.00	18.95	6.00	.300
Do	13.50	17.05	5.75	.288

Average height of flame, 18.74 inches.

Average duration of flame, 0.321 millisecond.

IMPACT TEST.

Date, January 31, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
18	1	No explosion.	22	1	Explosion.
22	1	Do.	20	1	No explosion.
24	1	Explosion.	21	5	Do.
23	1	Do.			

The maximum height at which no explosion occurs established at 21 centimeters (8.27 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 167 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 25	4	Did not explode.	Jan. 26	3	Exploded.
Do	2	Exploded.	Jan. 28	4	Did not explode.
Jan. 26	3	Did not explode.	Do	4	Do.
Do	3	Do.			

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 3	[b]218	1.03	41.50	103.75	A	
Do	218	1.03	32.20	100.62	A	101.18
Do	218	1.03	31.80	99.38	A	
Jan. 23	[b]218	1.03	38.60	96.50	B	
Do	218	1.03	31.60	98.75	B	98.21
Do	218	1.03	31.80	99.38	B	
Do	218	1.03	29.50	92.19	C	
Do	218	1.03	29.80	93.12	C	92.50
Jan. 24	218	1.03	29.50	92.19	C	

[a] Including 18 grams of the original wrapper.
[b] Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=111.93$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.03$. $W=218$ grams.

$M=\frac{VPS}{W}=7{,}933$ kilograms per square centimeter (112,840 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, September 9, 1910.	Grams.
Solid	11.5
Liquid (water)	81.5
Gaseous	112.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 6	81.55	1.320	1,119.5
Do	81.55	1.310	1,111.0
Do	81.55	1.310	1,111.0

[a] Including 9 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,113.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 24	63.50	50.25	13.25
Do	63.50	50.75	12.75
Do	63.50	50.75	12.75

Average compression, 12.9 millimeters (0.51 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Jan. 27	63	258	195	15
Do	63	250	187	15
Feb. 15	63	257	194	15

Average expansion of bore hole, 192 cubic centimeters (11.71 cubic inches).

COLLIER POWDER NO. 6-L. F

Explosive, Collier powder No. 6-L. F.
Class 4, nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8½ inches.
Average weight, 170 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.02.
Color of explosive, corn.
Consistency, fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 26, 1910.
Unit swing on this date, 3.32 inches.
Weight of charge, in grams, 308, 308, 308.
Swing in inches, 3.30, 3.20, 3.20.
Average swing, in inches, 3.23.

3.23 : 3.32 : : 308 : (317).

Therefore the unit deflective charge of Collier powder No. 6–L. F. is 317 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Sept. 23	317	8.35	No ignition.	Sept. 29	317	.....	No ignition.
Do	317	8.52	Do.	Do	317	.....	Do.
Do	317	8.31	Do.	Do	317	.....	Do.
Do	317	8.31	Do.	Do	317	.....	Do.
Do	317	8.43	Do.	Do	317	.....	Do.
Do	317	8.43	Do.	Do	317	.....	Do.
Do	317	8.43	Do.	Do	317	.....	Do.
Do	317	8.23	Do.	TEST 4.			
Do	317	8.33	Do.				
Do	317	8.33	Do.	Sept. 30	[a] 680	4.40	No ignition.
TEST 3.				Do	[a] 680	4.20	Do.
				Do	[a] 680	4.30	Do.
Sept. 29	317	.....	No ignition.	Do	[a] 680	4.45	Do.
Do	317	.....	Do.	Do	[a] 680	4.40	Do.
Do	317	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 3	14.78	43.5	2,942
Do	14.44	43.5	3,012
Oct. 4	15.12	43.5	2,877

Average rate of detonation, 2,944 meters (9,660 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 21	11.75	14.84	5.75	0.288
Do	9.75	12.32	4.25	.212
Do	12.00	15.16	5.00	.250

Average height of flame, 14.11 inches.
Average duration of flame, 0.250 millisecond.

IMPACT TEST.

Date, February 24, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	31	1	No explosion.
30	1	No explosion.	31	1	Explosion.
32	1	Explosion.	30	5	No explosion.

The maximum height at which no explosion occurs established at 30 centimeters (11.81 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 164 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 27	6	Did not explode.	Feb. 28	2	Exploded.
Do	5	Do.	Do	3	Did not explode.
Do	3	Do.	Do	3	Do.
Do	1	Exploded.			

The minimum distance at which no explosion occurs established at 3 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 23	215	1.02	24.50	61.25	A	61.04
Do	215	1.02	24.50	61.25	A	
Do	215	1.02	24.25	60.62	A	
Mar. 17	[b] 215	1.02	19.00	59.38	B	58.33
Do	215	1.02	23.25	58.12	B	
Do	215	1.02	23.00	57.50	B	
Do	215	1.02	22.50	56.25	C	56.04
Mar. 13	215	1.02	22.25	55.62	C	
Do	215	1.02	22.50	56.25	C	

[a] Including 15 grams of the original wrapper.
[b] Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

P=1.911A+0.5B−1.411C=66.75 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.02. W=215 grams.

$M=\frac{VPS}{W}$=4,750 kilograms per square centimeter (67,560 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, March 1, 1911.	Grams.
Solid	59.1
Liquid (water)	11.2
Gaseous	130.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 6	80.95	0.667	566.0
Mar. 9	80.95	.677	574.5
Do	80.95	.678	575.4

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 572.0.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 20	63.75	51.50	12.25
Do	64.00	51.00	13.00
Do	64.00	51.50	12.50

Average compression, 12.6 millimeters (0.50 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 22	63	194	131	15
Do	63	202	139	15
Do	63	197	134	15

Average expansion of bore hole, 135 cubic centimeters (8.24 cubic inches).

COLLIER NO. 9.

Explosive, Collier No. 9.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 192 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.15.
Color of explosive, white and corn.
Consistency, granular and powdered, very fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 20, 1911.
Unit swing on this date, 3.06 inches.
Weight of charge, in grams, 270, 270, 270.
Swing, in inches, 2.95, 2.90, 2.90.
Average swing, in inches, 2.92.

2.92 : 3.06 : : 270 : (283).

Therefore the unit deflective charge of Collier No. 9 is 283 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Feb. 21	283	8.05	No ignition.	Feb. 23	283	.....	No ignition.
Do	283	7.86	Do.	Do	283	.....	Do.
Do	283	7.84	Do.	Do	283	.....	Do.
Do	283	7.86	Do.	Do	283	.....	Do.
Do	283	7.90	Do.	Do	283	.....	Do.
Do	283	7.81	Do.	Do	283	.....	Do.
Do	283	7.94	Do.	Do	283	.....	Do.
Do	283	7.88	Do.	TEST 4.			
Do	283	8.00	Do.				
Do	283	7.99	Do.	Feb. 23	[a] 680	3.98	No ignition.
				Do	[a] 680	3.95	Do.
TEST 3.				Do	[a] 680	3.93	Do.
Feb. 23	283	.....	No ignition.	Feb. 24	[a] 680	3.87	Do.
Do	283	.....	Do.	Do	[a] 680	3.94	Do.
Do	283	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 20	21.55	45.00	2,068
Feb. 21	22.23	45.25	2,036
Do	22.55	45.00	1,996

Average rate of detonation, 2,040 meters (6,690 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 30	3.50	4.42	1.00	0.050
Do	9.50	12.00	3.00	.150
Do	7.25	9.16	2.25	.112

Average height of flame, 8.53 inches.
Average duration of flame, 0.104 millisecond.

IMPACT TEST.

Date October 18, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	31	1	No explosion.
36	1	Do.	31	1	Explosion.
34	1	Do.	30	5	No explosion.
32	1	Do.			

The maximum height at which no explosion occurs established at 30 centimeters (11.81 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 187 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 4	3	Exploded.	Nov. 6	8	Exploded.
Do	4	Do.	Do	9	Do.
Do	5	Did not explode.	Nov. 7	10	Did not explode.
Nov. 6	5	Exploded.	Do	10	Do.
Do	6	Do.	Do	10	Do.
Do	7	Do.			

The minimum distance at which no explosion occurs established at 10 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 8	[b] 212	1.15	23.75	74.22	A	73.44
Do	[b] 212	1.15	23.50	73.44	A	
Do	[b] 212	1.15	23.25	72.66	A	
Do	212	1.15	27.00	67.50	B	68.54
Aug. 9	212	1.15	28.25	70.62	B	
Do	212	1.15	27.00	67.50	B	
Aug. 10	212	1.15	26.50	66.25	C	67.29
Do	212	1.15	26.50	66.25	C	
Do	212	1.15	27.75	69.38	C	

[a] Including 12 grams of the original wrapper.
[b] Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

P=1.911A+0.5B−1.411C=79.67 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.15. W=212 grams.

$M=\frac{VPS}{W}$=6,483 kilograms per square centimeter (92,220 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 12 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 20, 1911.

	Grams.
Solid	35.8
Liquid (water)	58.0
Gaseous	113.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 28	80.95	0.833	718.8
July 29	80.95	.812	700.5
Do	80.95	.814	702.2

[a] Including 6 grams of the original wrapper.

Average large calories per kilogram of explosive, 707.2.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Oct. 23	63.25	51.75	11.50
Do	63.50	51.50	12.00
Oct. 24	64.00	52.50	11.50

Average compression, 11.7 millimeters (0.46 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 8	63	192	129	15
Do	63	192	129	15
Do	63	200	137	15

Average expansion of bore hole, 132 cubic centimeters (8.05 cubic inches).

COLLIER POWDER NO. 11.

Explosive, Collier powder No. 11.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Keystone National Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 174 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, drab.
Consistency, granular and fibrous, fine, dry, soft, moderately cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, January 2, 1913.

Unit swing on this date, 3.40 inches.

Weight of charge, in grams, 220, 220, 220.

Swing, in inches, 3.40, 3.53, 3.41.

Average swing, in inches, 3.45.

3.45 : 3.40 : : 220 : (217).

Therefore the unit deflective charge of Collier powder No. 11 is 217 grams.

GAS AND DUST GALLERY No. 1.

Date (1913).	Weight of charge.	Methane and ethane.	Result.	Date (1913).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 6	217	7.91	No ignition.	Jan. 10	217	.....	No ignition.
Do	217	7.90	Do.	Do	217	.....	Do.
Do	217	8.21	Do.	Do	217	.....	Do.
Do	217	7.91	Do.	Do	217	.....	Do.
Do	217	8.20	Do.	Do	217	.....	Do.
Do	217	7.75	Do.	Do	217	.....	Do.
Do	217	8.13	Do.	Do	217	.....	Do.
Do	217	7.92	Do.				
Do	217	7.91	Do.	TEST 4.			
Do	217	7.97	Do.				
				Jan. 8	[a]680	4.07	No ignition.
TEST 3.				Jan. 9	[a]680	4.00	Do.
				Do	[a]680	3.99	Do.
Jan. 10	217	.....	No ignition.	Do	[a]680	3.93	Do.
Do	217	.....	Do.	Do	[a]680	4.15	Do.
Do	217	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1913).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 6	11.65	42	3,605
Jan. 7	11.80	42	3,559
Jan. 8	11.65	41	3,519

Average rate of detonation, 3,561 meters (11,680 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 21	8.00	10.11	5.50	0.225
Do	9.00	11.37	6.25	.312
Do	8.75	11.05	6.75	.338

Average height of flame, 10.84 inches.

Average duration of flame, 0.292 millisecond.

IMPACT TEST.

Date, March 4, 1913.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	22	1	Explosion.
36	1	Do.	18	1	Do.
32	1	Do.	14	1	No explosion.
28	1	Do.	16	1	Do.
26	1	Do.	17	5	Do.

The maximum height at which no explosion occurs established at 17 centimeters (6.69 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 167 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 27	4	Exploded.	Feb. 27	9	Exploded.
Do	6	Do.	Do	10	Did not explode.
Do	8	Do.	Do	10	Do.
Do	10	Did not explode.			

The minimum distance at which no explosion occurs established at 10 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 14	215	1.04	25.00	104.17	A	
Do	215	1.04	24.75	103.12	A	103.82
Feb. 15	215	1.04	25.00	104.17	A	
Feb. 14	215	1.04	24.75	103.12	B	
Do	215	1.04	25.00	104.17	B	102.78
Feb. 17	215	1.04	24.25	101.04	B	
Do	215	1.04	23.75	98.96	C	
Do	215	1.04	24.75	103.12	C	101.73
Do	215	1.04	24.75	103.12	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=106.25$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.04$. $W=215$ grams.

$M=\frac{VPS}{W}=7{,}709$ kilograms per square centimeter (109,650 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, January 16, 1913.	Grams.
Solid	4.7
Liquid (water)	69.0
Gaseous	129.2

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 17	81.95	1.208	1,042.7
Do	81.95	1.208	1,042.7
Do	81.95	1.219	1,052.3

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,045.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 25	63.50	46.00	17.50
Do	63.25	46.00	17.25
Do	63.25	46.25	17.00

Average compression, 17.2 millimeters (0.67 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1913).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 6	63	310	247	15
Do	63	300	237	15
Do	63	307	244	15

Average expansion of bore hole, 243 cubic centimeters (14.82 cubic inches).

CRONITE NO. 1.

Explosive, Cronite No. 1.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 155 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, straw.
Consistency, granular, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, October 28, 1912.

Unit swing on this date, 3.43 inches.

Weight of charge, in grams, 240, 240, 240.

Swing, in inches, 3.54, 3.48, 3.58.

Average swing, in inches, 3.53.

3.53 : 3.43 : : 240 : (233).

Therefore the unit deflective charge of Cronite No. 1 is 233 grams.

GAS AND DUST GALLERY No. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Oct. 29	233	8.17	No ignition.	Nov. 4	233	.....	No ignition.
Do	233	8.34	Do.	Do	233	.....	Do.
Do	233	8.03	Do.	Do	233	.....	Do.
Do	233	8.18	Do.	Do	233	.....	Do.
Do	233	8.34	Do.	Do	233	.....	Do.
Do	233	7.78	Do.	Do	233	.....	Do.
Do	233	8.45	Do.	Nov. 5	233	.....	Do.
Do	233	8.28	Do.				
Oct. 30	233	7.95	Do.	TEST 4.			
Do	233	8.52	Do.				
				Oct. 31	a 680	4.13	No ignition.
TEST 3.				Do	a 680	4.19	Do.
				Nov. 1	a 680	4.21	Do.
Nov. 4	233	.....	No ignition.	Do	a 680	4.17	Do.
Do	233	.....	Do.	Do	a 680	4.26	Do.
Do	233	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 28	13.20	42.0	3,182
Do	13.15	42.0	3,194
Do	13.05	42.0	3,218

Average rate of detonation, 3,198 meters (10,490 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	10.75	13.58	6.00	0.300
Do	10.75	13.58	6.00	.300
Do	9.75	12.32	6.50	.325

Average height of flame, 13.16 inches.

Average duration of flame, 0.308 millisecond.

Impact Test.

Date, November 7, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
10	1	No explosion.	26	1	Explosion.
20	1	Do.	25	1	No explosion.
30	1	Explosion.	25	1	Explosion.
24	1	No explosion.	24	1	Do.
27	1	Do.	28	2	No explosion.
29	4	Do.	23	1	Explosion.
29	1	Explosion.	22	1	Do.
28	3	No explosion.	21	3	No explosion.
28	1	Explosion.	21	1	Explosion.
27	1	Do.	20	1	Do.
26	1	No explosion.	19	5	No explosion.

The maximum height at which no explosion occurs established at 19 centimeters (7.48 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 150 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 11.....	2	Exploded.	Nov. 11.....	5	Exploded.
Do.....	3	Do.	Nov. 16.....	6	Did not explode.
Do.....	4	Did not explode.	Do.....	6	Do.
Do.....	4	Exploded.	Do.....	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 29.....	215	0.93	23.75	98.96	A	97.57
Do.....	215	.93	23.75	98.96	A	
Do.....	215	.93	22.75	94.79	A	
Nov. 27	[b] 215	.93	31.50	98.44	B	95.57
Do.....	[b] 215	.93	30.00	93.75	B	
Do.....	[b] 215	.93	30.25	94.53	B	
Do.....	215	.93	20.50	85.42	C	84.73
Do.....	215	.93	20.25	84.38	C	
Do.....	215	.93	20.25	84.38	C	

[a] Including 15 grams of the original wrapper.
[b] Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=114.69$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.93$. $W=215$ grams.

$M=\frac{VPS}{W}=7{,}442$ kilograms per square centimeter (105,850 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 5, 1912.	Grams.
Solid	21.0
Liquid (water)	59.5
Gaseous	126.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 17	81.95	1.116	992.7
Do	81.95	1.151	993.1
Do	81.95	1.106	954.0

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 969.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 25	63.50	49.25	14.25
Do	63.50	49.00	14.50
Do	62.75	48.75	14.00

Average compression, 14.2 millimeters (0.56 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 14	62	338	276	15
Do	62	343	281	15
Do	62	351	289	15

Average expansion of bore hole, 282 cubic centimeters (17.20 cubic inches).

CRONITE NO. 5.

Explosive, Cronite No. 5.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 152 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, not cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, October 28, 1912.
Unit swing on this date, 3.43 inches.
Weight of charge, in grams, 255, 255, 255.
Swing, in inches, 3.43, 3.57, 3.53.
Average swing, in inches, 3.51.

3.51 : 3.43 : : 255 : (249).

Therefore the unit deflective charge of Cronite No. 5 is 249 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 30	249	7.83	No ignition.	Nov. 5	249	.....	No ignition.
Do	249	8.09	Do.	Do	249	.....	Do.
Do	249	7.92	Do.	Do	249	.....	Do.
Do	249	7.77	Do.	Do	249	.....	Do.
Do	249	7.84	Do.	Do	249	.....	Do.
Do	249	7.98	Do.	Do	249	.....	Do.
Do	249	8.06	Do.	Do	249	.....	Do.
Do	249	7.91	Do.				
Do	249	8.15	Do.				
Do	249	8.09	Do.	TEST 4.			
				Nov. 1	a 680	4.12	No ignition.
TEST 3.				Do	a 680	4.05	Do.
				Do	a 680	4.17	Do.
Nov. 5	249	.....	No ignition.	Do	a 680	4.06	Do.
Do	249	.....	Do.	Do	a 680	4.00	Do.
Do	249	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 30	16.70	42	2,515
Do	17.06	42	2,463
Nov. 1	17.35	42	2,421

Average rate of detonation, 2,466 meters (8,090 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 24	12.50	15.79	6.25	0.312
Do	10.50	13.26	5.50	.275
Do	11.25	14.21	4.75	.238

Average height of flame, 14.42 inches.
Average duration of flame, 0.275 millisecond.

IMPACT TEST.

Date, November 8, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	Explosion.	18	1	No explosion.
27	1	Do.	19	1	Explosion.
10	1	No explosion.	18	4	No explosion.
16	1	Do.			

The maximum height at which no explosion occurs established at 18 centimeters (7.09 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 150 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 11	4	Did not explode.	Nov. 11	2	Did not explode.
Do	2	Do.	Do	2	Do.
Do	1	Exploded.			

The minimum distance at which no explosion occurs established at 2 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 29	219	0.93	20.50	85.42	A	
Do	219	.93	20.25	84.38	A	85.07
Do	219	.93	20.50	85.42	A	
Nov. 30	219	.93	19.75	82.29	B	
Do	219	.93	20.50	85.42	B	83.33
Do	219	.93	19.75	82.29	B	
Do	219	.93	19.75	82.29	C	
Dec. 2	219	.93	19.00	79.17	C	81.25
Do	219	.93	19.75	82.29	C	

[a] Including 19 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=89.59$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=0.93$. $W=219$ grams.
$M=\frac{VPS}{W}=5{,}707$ kilograms per square centimeter (81,170 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 19 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 5, 1912.	Grams.
Solid	25.7
Liquid (water)	61.5
Gaseous	126.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 7	81.95	1.069	904.9
Do	81.95	1.059	896.4
Do	81.95	1.085	918.6

[a] Including 9.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 906.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 24	63.50	50.75	12.75
Do	63.50	51.00	12.50
Do	63.50	51.25	12.25

Average compression, 12.5 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 14	62	286	224	15
Do	62	286	224	15
Do	62	285	223	15

Average expansion of bore hole, 224 cubic centimeters (13.66 cubic inches).

DETONITE SPECIAL.

Explosive, Detonite special.
Class 1*a*, ammonium nitrate.
Manufactured by the King Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 12 inches.
Average weight, 441 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.09.
Color of explosive, drab.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, February 1, 1910.
Unit swing on this date, 3.14 inches.
Weight of charge, in grams, 300, 300, 300.
Swing, in inches, 3.02, 3.03, 3.00.
Average swing, in inches, 3.02.

3.02 : 3.14 : : 300 : (312).

Therefore the unit deflective charge of Detonite special is 312 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 1—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 1	312	8.07	No ignition.	Feb. 2	312	.....	No ignition.
Do	312	8.02	Do.	Do	312	.....	Do.
Do	312	8.42	Do.	Do	312	.....	Do.
Feb. 2	312	8.07	Do.	Feb. 3	312	.....	Do.
Do	312	8.17	Do.	Do	312	.....	Do.
Do	312	8.27	Do.	Do	312	.....	Do.
Do	312	8.17	Do.	Do	312	.....	Do.
Do	312	8.22	Do.				
Do	312	8.17	Do.	TEST 4.			
Do	312	8.27	Do.				
				Jan. 31	[a] 680	3.96	No ignition.
TEST 3.				Do	[a] 680	4.07	Do.
				Do	[a] 680	4.15	Do.
Feb. 2	312	.....	No ignition.	Do	[a] 680	4.06	Do.
Do	312	.....	Do.	Do	[a] 680	3.84	Do.
Do	312	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 2	13.05	43.0	3,295
Do	13.11	43.0	3,280
Do	12.87	43.0	3,341

Average rate of detonation, 3,305 meters (10,840 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 30	5.00	6.32	1.50	0.075
Do	5.50	6.95	2.50	.125
Do	5.00	6.32	1.50	.075

Average height of flame, 6.53 inches.

Average duration of flame, 0.092 millisecond.

Impact Test.

Date, November 11, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
8	1	No explosion.	18	1	Explosion.
16	1	Do.	12	1	No explosion.
32	1	Explosion.	14	1	Explosion.
24	1	No explosion.	13	1	Do.
28	1	Explosion.	12	2	No explosion.
26	1	Do.	12	1	Explosion.
25	1	Do.	11	1	Do.
24	1	Do.	10	5	No explosion.
20	1	Do.			

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 206 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 17	3	Did not explode.	Dec. 17	2	Exploded.
Do	2	Do.	Dec. 18	3	Did not explode.
Do	1	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Dec. 4	238	0.84	26.50	82.81	A	82.29
Do	238	.84	26.50	82.81	A	
Do	238	.84	26.00	81.25	A	
Do	238	.84	25.25	78.91	B	79.96
Do	238	.84	25.50	79.69	B	
Do	238	.84	26.00	81.25	B	
Do	238	.84	23.25	72.66	C	73.18
Do	238	.84	23.00	71.88	C	
Do	238	.84	24.00	75.00	C	

[a] Including 38 grams of the original wrapper.

P=1.911A+0.5B−1.411C=93.97 kilograms per square centimeter.
V=15,000 cubic centimeters. S=0.84. W=238 grams.
$M=\frac{VPS}{W}$=4,975 kilograms per square centimeter (70,760 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 33 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 7, 1910.	Grams.
Solid	44.8
Liquid (water)	56.5
Gaseous	133.9

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 119.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 9	81.95	0.912	709.3
Dec. 10	81.95	.930	723.5
Do	81.95	.940	731.3

[a] Including 19 grams of the original wrapper.

Average large calories per kilogram of explosive, 721.4.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 9	63.50	52.25	11.25
Do	63.50	51.50	12.00
Do	63.50	51.00	12.50

Average compression, 11.9 millimeters (0.47 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 15	63	222	159	15
Do	63	227	164	15
Nov. 18	63	219	156	15

Average expansion of bore hole, 160 cubic centimeters (9.76 cubic inches).

EUREKA NO. 2.

Explosive, Eureka No. 2.
Class 2, hydrated.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 202 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.15.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, April 15, 1912.
Unit swing on this date, 3.37 inches.
Weight of charge, in grams, 295, 295, 295.
Swing, in inches, 3.22, 3.20, 3.17.
Average swing, in inches, 3.20.

3.20 : 3.37 : : 295 : (311).

Therefore the unit deflective charge of Eureka No. 2 is 311 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Apr. 19	311	8.15	No ignition.	Apr. 25	311	.....	No ignition.
Do	311	8.14	Do.	Do	311	.....	Do.
Do	311	7.95	Do.	Do	311	.....	Do.
Do	311	8.02	Do.	Do	311	.....	Do.
Do	311	8.09	Do.	Do	311	.....	Do.
Do	311	7.97	Do.	Do	311	.....	Do.
Apr. 20	311	7.98	Do.	Do	311	.....	Do.
Do	311	8.26	Do.				
Do	311	7.95	Do.	TEST 4.			
Do	311	7.94	Do.				
				Apr. 22	[a] 680	4.12	No ignition.
TEST 3.				Do	[a] 680	4.16	Do.
				Do	[a] 680	4.07	Do.
Apr. 25	311	.....	No ignition.	Do	[a] 680	4.17	Do.
Do	311	.....	Do.	Do	[a] 680	4.12	Do.
Do	311	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
May 3	13.70	44.0	3,212
Do	13.75	44.5	3,236
Do	13.10	44.0	3,359

Average rate of detonation, 3,269 meters (10,720 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Aug. 28	8.00	10.11	3.00	0.150
Do	7.75	9.79	3.00	.150
Do	10.75	13.58	4.50	.225

Average height of flame, 11.16 inches.
Average duration of flame, 0.175 millisecond.

Impact Test.

Date, August 20, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	16	1	Explosion.
30	1	Do.	15	1	Do.
20	1	Do.	14	1	Do.
15	1	No explosion.	13	5	No explosion.
18	1	Explosion.			

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 185 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 7	3	Exploded.	Sept. 7	7	Exploded.
Do	4	Do.	Do	8	Did not explode.
Do	5	Do.	Do	8	Do.
Do	6	Do.	Do	8	Do.

The minimum distance at which no explosion occurs established at 8 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 19	211	1.15	19.50	60.94	A	61.98
Do	211	1.15	20.00	62.50	A	
Aug. 20	211	1.15	20.00	62.50	A	
Do	211	1.15	18.25	57.03	B	57.03
Do	211	1.15	18.25	57.03	B	
Do	211	1.15	18.25	57.03	B	
Do	211	1.15	17.25	53.91	C	53.91
Aug. 21	211	1.15	17.25	53.91	C	
Do	211	1.15	17.25	53.91	C	

[a] Including 11 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=70.89$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.15$. $W=211$ grams.
$M=\frac{VPS}{W}=5{,}796$ kilograms per square centimeter (82,440 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 11 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, May 1, 1912.	Grams.
Solid	77.9
Liquid (water)	21.7
Gaseous	103.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 105.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 26	81.95	0.733	641.5
Do	81.95	.721	630.8
Do	81.95	.752	658.3

[a] Including 5.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 643.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 15	63.25	51.50	11.75
Do	63.25	51.50	11.75
Do	63.25	51.75	11.50

Average compression, 11.7 millimeters (0.46 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 27	62	181	119	15
Do	62	183	121	15
Do	62	184	122	15

Average expansion of bore hole, 121 cubic centimeters (7.38 cubic inches).

FORT PITT MINE POWDER NO. 1.

Explosive, Fort Pitt mine powder No. 1.
Class 4, nitroglycerin.
Manufactured by the Fort Pitt Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 173 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, March 21, 1912.
Unit swing on this date, 3.59 inches.
Weight of charge, in grams, 300, 300, 300.
Swing, in inches, 3.77, 3.67, 3.60.
Average swing, in inches, 3.68.

3.68 : 3.59 : : 300 : (293).

Therefore the unit deflective charge of Fort Pitt mine powder No. 1 is 293 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.			
	Grams.	*Per cent.*	
Mar. 21	293	7.77	No ignition.
Do	293	7.81	Do.
Mar. 22	293	7.96	Do.
Do	293	7.75	Do.
Do	293	8.02	Do.
Do	293	8.11	Do.
Do	293	8.00	Do.
Do	293	7.92	Do.
Do	293	8.33	Do.
Do	293	8.46	Do.
TEST 3.			
Mar. 25	293	.....	No ignition.
Do	293	.....	Do.
Do	293	.....	Do.
TEST 3—Con.			
	Grams.	*Per cent.*	
Mar. 25	293	.....	No ignition.
Do	293	.....	Do.
Do	293	.....	Do.
Do	293	.....	Do.
Do	293	.....	Do.
Do	293	.....	Do.
Do	293	.....	Do.
TEST 4.			
Mar. 25	a 680	4.21	No ignition.
Do	a 680	4.25	Do.
Mar. 26	a 680	3.93	Do.
Do	a 680	4.02	Do.
Do	a 680	3.99	Do.

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 25	13.05	44.0	3,372
Do	12.80	44.5	3,477
Do	13.00	44.5	3,423

Average rate of detonation, 3,424 meters (11,230 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Aug. 28	11.75	14.84	3.25	0.162
Do	11.50	14.53	4.00	.200
Do	9.50	12.00	3.50	.175

Average height of flame, 13.79 inches.
Average duration of flame, 0.179 millisecond.

Impact Test.

Date, May 17, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
25	1	Explosion.	10	1	No explosion.
20	1	Do.	13	1	Explosion.
15	1	Do.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 166 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 6	8	Did not explode.	Nov. 6	6	Did not explode.
Do	4	Exploded.	Do	6	Do.
Do	5	Do.	Do	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
July 26	212	1.04	21.75	67.97	A	68.23
July 27	212	1.04	22.00	68.75	A	
July 29	212	1.04	21.75	67.97	A	
July 24	212	1.04	21.00	65.62	B	66.15
Do	212	1.04	21.25	66.41	B	
Do	212	1.04	21.25	66.41	B	
July 26	212	1.04	20.00	62.50	C	63.54
Do	212	1.04	20.75	64.84	C	
Do	212	1.04	20.25	63.28	C	

[a] Including 12 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=73.81$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.04$. $W=212$ grams.
$M=\frac{VPS}{W}=5{,}431$ kilograms per square centimeter (77,250 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 12.6 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, March 22, 1912.	Grams.
Solid	62.3
Liquid (water)	11.2
Gaseous	129.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 10	81.95	0.925	804.0
Feb. 13	81.95	.895	777.7
Feb. 14	81.95	.934	811.9

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 797.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
May 18	63.25	50.00	13.25
Do	63.25	49.50	13.75
Do	63.00	49.75	13.25

Average compression, 13.4 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 27	62	235	173	15
Do	62	236	174	15
Do	62	228	166	15

Average expansion of bore hole, 172 cubic centimeters (10.49 cubic inches).

FUEL-ITE NO. 1.

Explosive, Fuel-ite No. 1.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 180 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.10.
Color of explosive, corn.
Consistency, fibrous, fine, dry, very soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, June 19, 1911.
Unit swing on this date, 2.98 inches.
Weight of charge, in grams, 280, 280, 280.
Swing, in inches, 2.97, 2.86, 2.86.
Average swing, in inches, 2.90.

$$2.90 : 2.98 : : 280 : (288).$$

Therefore the unit deflective charge of Fuel-ite No. 1 is 288 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
June 20	288	8.01	No ignition.	June 24	288	.....	No ignition.
June 21	288	7.87	Do.	Do	288	.....	Do.
Do	288	7.85	Do.	Do	288	.....	Do.
Do	288	8.18	Do.	Do	288	.....	Do.
Do	288	8.26	Do.	Do	288	.....	Do.
Do	288	7.96	Do.	Do	288	.....	Do.
Do	288	8.12	Do.	Do	288	.....	Do.
Do	288	8.05	Do.	Do	288	.....	Do.
Do	288	8.10	Do.				
June 30	288	8.46	Do.	TEST 4.			
TEST 3.				June 28	a 680	3.95	No ignition.
				Do	a 680	3.82	Do.
June 24	288	.....	No ignition.	Do	a 680	3.95	Do.
Do	288	.....	Do.	Do	a 680	3.87	Do.
Do	288	.....	Do.	June 29	a 680	3.87	Do.

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 13	14.04	48.00	3,419
June 14	15.05	51.75	3,439
Do	15.26	51.00	3,342

Average rate of detonation, 3,400 meters (11,150 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 2	21.75	27.47	10.50	0.525
Do	16.25	20.53	8.25	.413
Do	23.50	29.69	12.75	.638

Average height of flame, 25.89 inches.
Average duration of flame, 0.525 millisecond.

Impact Test.

Date, December 28, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	18	1	Explosion.
12	1	No explosion.	16	2	No explosion.
13	1	Do.	16	1	Explosion.
14	1	Do.	15	5	No explosion.

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 180 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 19	2	Exploded.	Jan. 19	5	Did not explode.
Do	3	Do.	Do	5	Do.
Do	4	Did not explode.	Do	5	Do.
Do	4	Exploded.			

The minimum distance at which no explosion occurs established at 5 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 21	217	1.10	22.25	69.53	A	70.83
Do	217	1.10	22.50	70.31	A	
Do	217	1.10	23.25	72.66	A	
Nov. 20	217	1.10	21.50	67.19	B	67.71
Nov. 21	217	1.10	21.75	67.97	B	
Do	217	1.10	21.75	67.97	B	
Nov. 16	217	1.10	20.00	62.50	C	63.28
Do	217	1.10	20.00	62.50	C	
Nov. 17	217	1.10	20.75	64.84	C	

[a] Including 17 grams of the original wrapper.

P=1.911A+0.5B−1.411C=79.92 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.10. W=217 grams.
$M=\frac{VPS}{W}$=6,077 kilograms per square centimeter (86,440 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 12, 1911.	Grams.
Solid	70.6
Liquid (water)	10.6
Gaseous	126.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 23	81.95	0.808	688.4
Do	81.95	.822	700.5
Jan. 24	81.95	.782	666.0

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 685.0.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Nov. 21	63.50	49.25	14.25
Do	63.50	49.00	14.50
Do	63.50	49.00	14.50

Average compression, 14.4 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 13	63	228	165	15
Do	63	226	163	15
Do	63	228	165	15

Average expansion of bore hole, 164 cubic centimeters (10.0 cubic inches).

FUEL-ITE NO. 2.

Explosive, Fuel-ite No. 2.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 168 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.02.
Color of explosive, corn.
Consistency, fibrous, fine, dry, very soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, June 19, 1911.
Unit swing on this date, 2.98 inches.
Weight of charge, in grams, 305, 305, 305.
Swing, in inches, 3.12, 3.13, 3.17.
Average swing, in inches, 3.14.

3.14 : 2.98 : : 305 : (289).

Therefore the unit deflective charge of Fuel-ite No. 2 is 289 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 22	289	7.97	No ignition.	June 26	289	.....	No ignition.
Do	289	8.04	Do.	Do	289	.....	Do.
Do	289	8.23	Do.	Do	289	.....	Do.
Do	289	8.15	Do.	Do	289	.....	Do.
Do	289	8.04	Do.	Do	289	.....	Do.
Do	289	8.19	Do.	Do	289	.....	Do.
Do	289	8.27	Do.	Do	289	.....	Do.
Do	289	7.96	Do.				
Do	289	8.01	Do.	TEST 4.			
July 31	289	7.89	Do.				
				June 27	a680	4.02	No ignition.
TEST 3.				Do	a680	3.94	Do.
				Do	a680	3.92	Do.
June 26	289	.....	No ignition.	Do	a680	3.93	Do.
Do	289	.....	Do.	Do	a680	3.82	Do.
Do	289	.....	Do.				

a680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 19	17.54	43.50	2,480
Do	17.24	43.50	2,523
Do	17.18	43.50	2,532

Average rate of detonation, 2,512 meters (8,240 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 11	8.75	11.05	4.00	0.200
Do	8.75	11.05	3.75	.188
Do	8.00	10.10	3.75	.188

Average height of flame, 10.73 inches.
Average duration of flame, 0.192 millisecond.

Impact Test.

Date, December 28, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	1	Explosion.
18	1	Do.	14	3	No explosion.
16	1	No explosion.	14	1	Explosion.
17	1	Explosion.	13	5	No explosion.
16	1	Do.			

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 168 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 19	4	Did not explode.	Jan. 19	1	Exploded.
Do	3	Do.	Do	2	Did not explode.
Do	2	Do.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 22	217	1.02	20.00	62.50	A	61.98
Do	217	1.02	20.00	62.50	A	
Do	217	1.02	19.50	60.94	A	
Nov. 23	217	1.02	19.50	60.94	B	59.90
Do	217	1.02	18.75	58.59	B	
Nov. 24	217	1.02	19.25	60.16	B	
Do	217	1.02	18.50	57.81	C	56.51
Do	217	1.02	18.00	56.25	C	
Do	217	1.02	17.75	55.47	C	

[a] Including 17 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=68.66$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.02$. $W=217$ grams.
$M=\frac{VPS}{W}=4{,}841$ kilograms per square centimeter (68,860 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 13, 1911.	Grams.
Solid	80.9
Liquid (water)	8.2
Gaseous	123.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 25	81.95	0.809	669.2
Do	81.95	.789	672.0
Do	81.95	.788	671.2

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 677.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Nov. 21	63.50	53.50	10.00
Do	63.25	53.50	9.75
Do	63.50	53.75	9.75

Average compression, 9.8 millimeters (0.39 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 13	63	196	133	15
Do	63	199	136	15
Do	63	200	137	15

Average expansion of bore hole, 135 cubic centimeters (8.24 cubic inches).

FUEL-ITE NO. 3.

Explosive, Fuel-ite No. 3.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 145 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.91.
Color of explosive, corn.
Consistency, fibrous, fine, dry, very soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, June 19, 1911.
Unit swing on this date, 2.98 inches.
Weight of charge, in grams, 225, 225, 225.
Swing, in inches, 3.05, 3.17, 3.14.
Average swing, in inches, 3.12.

3.12 : 2.98 : : 225 : (215).

Therefore the unit deflective charge of Fuel-ite No. 3 is 215 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 23	215	8.05	No ignition.	June 26	215	.....	No ignition.
Do	215	8.05	Do.	Do	215	.....	Do.
Do	215	8.02	Do.	Do	215	.....	Do.
Do	215	8.25	Do.	Do	215	.....	Do.
Do	215	8.00	Do.	Do	215	.....	Do.
Do	215	7.82	Do.	Do	215	.....	Do.
Do	215	8.07	Do.	Do	215	.....	Do.
Do	215	8.14	Do.				
Do	215	7.94	Do.	TEST 4.			
Do	215	7.85	Do.				
				June 29	[a]680	3.88	No ignition.
TEST 3.				Do	[a]680	3.84	Do.
				Do	[a]680	3.81	Do.
June 26	215	.....	No ignition.	Do	[a]680	4.12	Do.
Do	215	.....	Do.	Do	[a]680	3.87	Do.
Do	215	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 19	16.53	43.50	2,585
Do	16.65	43.50	2,613
Do	16.35	43.50	2,661

Average rate of detonation, 2,620 meters (8,590 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 2	14.00	17.66	5.25	0.262
Do	18.25	23.05	7.00	.350
Do	18.75	23.68	7.00	.350

Average height of flame, 21.47 inches.
Average duration of flame, 0.321 millisecond.

Impact Test.

Date, January 10, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	23	1	Explosion.
30	1	Explosion.	22	5	No explosion.
25	1	Do.			

The maximum height at which no explosion occurs established at 22 centimeters (8.66 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 146 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 8	2	Did not explode.	Feb. 8	2	Did not explode.
Do	1	Exploded.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Nov. 29	220	0.91	31.25	97.64	A	
Do	220	.91	31.00	96.87	A	96.09
Do	220	.91	30.00	93.75	A	
Dec. 1	220	.91	29.50	92.19	B	
Do	220	.91	29.75	92.97	B	91.98
Do	220	.91	29.00	90.63	B	
Do	220	.91	28.00	87.50	C	
Do	220	.91	27.50	85.93	C	86.98
Dec. 2	220	.91	28.00	87.50	C	

[a] Including 20 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=104.66$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=0.91$. $W=220$ grams.
$M=\frac{VPS}{W}=6{,}630$ kilograms per square centimeter (94,310 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 20 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 14, 1911.	Grams.
Solid	16.3
Liquid (water)	64.0
Gaseous	130.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 25	81.95	1.203	1,014.7
Jan. 26	81.95	1.210	1,020.7
Do	81.95	1.187	1,001.1

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,012.2.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Nov. 21	63.50	50.25	13.25
Do	63.50	50.00	13.50
Nov. 28	63.50	50.25	13.25

Average compression, 13.3 millimeters (0.52 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Jan. 16	62	346	284	18
Do	62	338	276	20
Do	62	348	286	16

Average expansion of bore hole, 282 cubic centimeters (17.20 cubic inches).

GIANT A LOW-FLAME DYNAMITE.

Explosive, Giant A low-flame dynamite.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 216 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.27.
Color of explosive, corn.
Consistency, granular, coarse, wet, hard, moderately cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, July 1, 1909.
Unit swing on this date, 2.89 inches.
Weight of charge, in grams, 350, 350, 350.
Swing, in inches, 3.04, 3.02, 2.94.
Average swing, in inches, 3.00.

3.00 : 2.89 : : 350 : (337).

Therefore the unit deflective charge of Giant A low-flame dynamite is 337 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 1	337	8.11	No ignition.	July 3	337	.....	No ignition.
Do	337	8.11	Do.	Do	337	.....	Do.
Do	337	8.06	Do.	Do	337	.....	Do.
July 2	337	8.11	Do.	Do	337	.....	Do.
Do	337	8.22	Do.	July 6	337	.....	Do.
Do	337	8.22	Do.	Do	337	.....	Do.
Do	337	7.97	Do.	Do	337	.....	Do.
Do	337	8.06	Do.				
Do	337	8.06	Do.	TEST 4.			
Do	337	8.06	Do.				
				July 1	[a] 680	3.95	No ignition.
TEST 3.				Do	[a] 680	4.02	Do.
				Do	[a] 680	3.95	Do.
July 3	337	.....	No ignition.	Do	[a] 680	4.02	Do.
Do	337	.....	Do.	Do	[a] 680	4.11	Do.
Do	337	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 25	17.30	43	2,486
Aug. 26	17.31	43	2,484
Sept. 1	17.38	43	2,474

Average rate of detonation, 2,481 meters (8,140 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 30	15.25	19.26	8.50	0.425
Do	14.50	18.32	6.25	.312
Do	16.50	20.84	9.25	.462

Average height of flame, 19.47 inches.

Average duration of flame, 0.400 millisecond.

Impact Test.

Date, May 2, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
10	1	No explosion.	18	1	No explosion.
20	1	Explosion.	19	5	Do.

The maximum height at which no explosion occurs established at 19 centimeters (7.48 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 214 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 31	6	Did not explode.	Feb. 12	9	Exploded.
Do	4	Do.	Do	14	Did not explode.
Do	3	Exploded.	Do	11	Exploded.
Do	4	Do.	Do	13	Did not explode.
Do	5	Do.	Do	12	Do.
Do	6	Do.	Do	12	Exploded.
Do	7	Do.	Do	13	Did not explode.
Feb. 12	8	Do.	Do	13	Do.

The minimum distance at which no explosion occurs established at 13 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 5	214	1.33	19.25	60.16	A	
Feb. 6	214	1.33	19.75	61.72	A	61.46
Do	214	1.33	20.00	62.50	A	
Feb. 5	214	1.33	18.75	58.59	B	
Do	214	1.33	18.75	58.59	B	58.59
Do	214	1.33	18.75	58.59	B	
Do	214	1.33	19.00	59.38	C	
Do	214	1.33	18.75	58.59	C	58.59
Do	214	1.33	18.50	57.81	C	

[a] Including 14 grams of the original wrapper.

P=1.911A+0.5B−1.411C=64.07 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.33. W=214 grams.

$M=\frac{VPS}{W}$=5,973 kilograms per square centimeter (84,960 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, February 7, 1913.	Grams.
Solid	83.4
Liquid (water)	17.1
Gaseous	95.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 3	81.95	0.772	666.5
Do	81.95	.801	691.9
Do	81.95	.764	659.5

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 672.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1909).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 7	64.00	52.50	11.50
Do	64.00	52.00	12.00
Do	64.00	52.50	11.50

Average compression, 11.7 millimeters (0.46 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCK.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 19	62	214	152	15
Do	62	210	148	15
Do	62	208	146	15

Average expansion of bore hole, 149 cubic centimeters (9.09 cubic inches).

GIANT B LOW-FLAME DYNAMITE.

Explosive, Giant B low-flame dynamite.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 237 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.35.
Color of explosive, corn.
Consistency, granular, coarse, wet, hard, moderately cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, July 1, 1909.
Unit swing on this date, 2.89.
Weight of charge, in grams, 350, 350, 350.
Swing, in inches, 2.77, 2.72, 2.76.
Average swing, in inches, 2.75.

2.75 : 2.89 : : 350 : (368).

Therefore the unit deflective charge of Giant B low-flame dynamite is 368 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 6	368	7.97	No ignition.	July 7	368	.....	No ignition.
Do	368	8.22	Do.	Do	368	.....	Do.
Do	368	8.06	Do.	Do	368	.....	Do.
Do	368	7.88	Do.	Do	368	.....	Do.
Do	368	7.88	Do.	Do	368	.....	Do.
Do	368	7.88	Do.	Do	368	.....	Do.
Do	368	7.79	Do.	Do	368	.....	Do.
Do	368	7.96	Do.				
Do	368	7.94	Do.	TEST 4.			
Do	368	7.79	Do.				
				July 6	a 680	3.95	No ignition.
TEST 3.				July 15	a 680	4.25	Do.
				Do	a 680	4.00	Do.
July 7	368	.....	No ignition.	Do	a 680	4.00	Do.
Do	368	.....	Do.	Do	a 680	4.17	Do.
Do	368	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 26	15.47	43.0	2,780
Do	15.48	43.0	2,778
Sept. 2	15.57	43.0	2,762

Average rate of detonation, 2,773 meters (9,100 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 29	16.00	20.21	7.50	0.375
Do	13.50	17.05	8.25	.412
Do	18.50	23.37	9.75	.488

Average height of flame, 20.21 inches.

Average duration of flame, 0.425 millisecond.

Impact Test.

Date, May 2, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	28	1	No explosion.
30	1	Explosion.	29	5	Do.
25	1	No explosion.			

The maximum height at which no explosion occurs established at 29 centimeters (11.42 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 224 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Jan. 31	6	Exploded.	Feb. 1	16	Exploded.
Do	10	Do.	Feb. 11	17	Did not explode.
Do	14	Do.	Do	17	Exploded.
Do	18	Did not explode.	Do	18	Did not explode.
Feb. 1	16	Do.	Do	18	Do.
Do	15	Exploded.			

The minimum distance at which no explosion occurs established at 18 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 6	213	1.39	17.75	55.47	A	55.99
Do	213	1.39	18.25	57.03	A	
Do	213	1.39	17.75	55.47	A	
Do	213	1.39	16.25	50.78	B	50.52
Do	213	1.39	16.00	50.00	B	
Feb. 7	213	1.39	16.25	50.78	B	
Do	213	1.39	15.75	49.22	C	49.22
Do	213	1.39	15.75	49.22	C	
Do	213	1.39	15.75	49.22	C	

[a] Including 13 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=62.81$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.39$. $W=213$ grams.
$M=\frac{VPS}{W}=6{,}148$ kilograms per square centimeter (87,440 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst A. L. Hyde.]

Date, February 10, 1913.	Grams.
Solid	61.8
Liquid (water)	36.1
Gaseous	91.5

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1909).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 7	64.00	51.80	12.20
Do	64.00	52.00	12.00
Do	64.00	52.00	12.00

Average compression, 12.1 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 19	62	200	138	15
Do	62	204	142	15
Do	62	200	138	15

Average expansion of bore hole, 139 cubic centimeters (8.48 cubic inches).

GIANT C LOW-FLAME DYNAMITE.

Explosive, Giant C low-flame dynamite.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 274 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.55.
Color of explosive, drab.
Consistency, granular, coarse, wet, soft, moderately cohesive.

Unit deflective charge as determined by the ballistic pendulum.

Date, July 2, 1909.

Unit swing on this date, 2.89.

Weight of charge, in grams, 310, 310, 310.

Swing, in inches, 3.04, 2.94, 2.99.

Average swing, in inches, 2.99.

$$2.99 : 2.89 :: 310 : (300).$$

Therefore the unit deflective charge of Giant C low-flame dynamite is 300 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
July 8	300	7.96	No ignition.	July 8	300	.....	No ignition.
Do	300	8.13	Do.	Do	300	.....	Do.
Do	300	7.96	Do	Do	300	.....	Do.
Do	300	7.96	Do.	Do	300	.....	Do.
Do	300	8.13	Do.	July 9	300	.....	Do.
Do	300	8.30	Do.	Do	300	.....	Do.
Do	300	7.96	Do.	Do	300	.....	Do.
Do	300	7.79	Do.				
Do	300	8.13	Do.	TEST 4.			
Do	300	7.96	Do.				
				July 8	[a]680	3.97	No ignition.
TEST 3.				July 15	[a]680	4.17	Do.
				Do	[a]680	4.17	Do.
July 8	300	.....	No ignition.	July 16	[a]680	4.17	Do.
Do	300	.....	Do.	Do	[a]680	4.09	Do.
Do	300	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 26	15.55	43.0	2,765
Aug. 27	16.05	43.0	2,679
Oct. 9	16.15	43.0	2,663

Average rate of detonation, 2,702 meters (8,860 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 29	18.50	23.37	9.50	0.475
Do	22.00	27.79	9.75	.488
Do	20.25	25.58	10.25	.512

Average height of flame, 25.58 inches.

Average duration of flame, 0.492 millisecond.

IMPACT TEST.

Date, May 3, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
28	1	No explosion.	32	1	Explosion.
30	1	Do.	31	1	Do.
34	1	Explosion.	30	1	Do.
32	1	No explosion.	28	1	Do.
33	1	Explosion.	27	1	Do.
32	1	No explosion.	26	5	No explosion.

The maximum height at which no explosion occurs established at 26 centimeters (10.24 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 240 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 12	15	Exploded.	Feb. 12	20	Did not explode
Do	17	Do.	Do	20	Do.
Do	19	Do.	Do	20	Do.
Do	21	Did not explode.			

The minimum distance at which no explosion occurs established at 20 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 6	212	1.49	21.75	67.97	A	
Do	212	1.49	21.00	65.62	A	66.93
Do	212	1.49	21.50	67.19	A	
Feb. 7	212	1.49	20.75	64.84	B	
Do	212	1.49	20.00	62.50	B	63.80
Do	212	1.49	20.50	64.06	B	
Do	212	1.49	19.00	59.38	C	
Do	212	1.49	19.00	59.38	C	59.64
Feb. 8	212	1.49	19.25	60.16	C	

[a] Including 12 grams of the original wrapper.

P=1.911A+0.5B−1.411C=75.65 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.49. W=212 grams.

$M=\frac{VPS}{W}$=7,975 kilograms per square centimeter (113,430 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 12 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, February 11, 1913.

	Grams.
Solid	64.6
Liquid (water)	23.2
Gaseous	94.7

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 5	81.95	0.947	827.2
Do	81.95	.955	834.2
Do	81.95	.913	797.3

[a] Including 6 grams of the original wrapper.

Average large calories per kilogram of explosive, 819.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1909).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 7	64.00	49.30	14.70
Do	64.00	49.80	14.20
Do	64.00	49.10	14.90

Average compression, 14.6 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	174	111	15
Do	63	177	114	15
Do	63	172	109	15

Average expansion of bore hole, 111 cubic centimeters (6.77 cubic inches).

GIANT COAL-MINE POWDER NO. 5.

Explosive, Giant coal-mine powder No. 5.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 157 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.89.
Color of explosive, corn.
Consistency, granular and fibrous, mostly very fine with large white crystals, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 12, 1911.

Unit swing on this date, 3.23 inches.

Weight of charge, in grams, 230, 230, 230.

Swing, in inches, 3.25, 3.22, 3.32.

Average swing, in inches, 3.26.

3.26 : 3.23 : : 230 : (228).

Therefore the unit deflective charge of Giant coal-mine powder No. 5 is 228 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 15	228	8.16	No ignition.	Oct. 5	228	.....	No ignition.
Do	228	8.27	Do.	Do	228	.....	Do.
Do	228	8.11	Do.	Do	228	.....	Do.
Sept. 16	228	8.19	Do.	Do	228	.....	Do.
Do	228	8.19	Do.	Do	228	.....	Do.
Do	228	8.31	Do.	Do	228	.....	Do.
Do	228	8.32	Do.	Do	228	.....	Do.
Sept. 18	228	7.97	Do.				
Do	228	8.06	Do.	TEST 4.			
Do	228	7.90	Do.				
				Sept. 13	[a] 680	4.19	No ignition.
TEST 3.				Do	[a] 680	4.26	Do.
				Sept. 15	[a] 680	4.06	Do.
Oct. 5	228	.....	No ignition.	Sept. 18	[a] 680	4.26	Do.
Do	228	.....	Do.	Do	[a] 680	4.23	Do.
Do	228	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 13	14.70	45.0	3,061
Sept. 14	14.10	45.0	3,191
Sept. 18	14.50	45.0	3,103

Average rate of detonation, 3,118 meters (10,230 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	16.50	20.84	6.50	0.325
Do	19.50	24.68	8.00	.400
Do	19.50	24.68	8.75	.438

Average height of flame, 23.37 inches.

Average duration of flame, 0.388 millisecond.

Impact Test.

Date, January 11, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
24	1	Explosion.	15	1	Explosion.
16	1	No explosion.	14	1	No explosion.
20	1	Explosion.	14	1	Explosion.
18	1	Do.	13	2	No explosion.
16	1	No explosion.	16	1	Explosion.
16	1	Explosion.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 147 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 28.....	2	Did not explode.	Feb. 28.....	2	Did not explode.
Do.....	1	Exploded.	Do.....	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 9.....	217	0.89	30.00	93.75	A	94.27
Do.....	217	.89	30.00	93.75	A	
Feb. 10.....	217	.89	30.50	95.31	A	
Feb. 2.....	217	.89	30.50	95.31	B	97.14
Do.....	217	.89	31.75	99.22	B	
Feb. 5.....	217	.89	31.00	96.88	B	
Do.....	217	.89	26.75	83.59	C	82.81
Do.....	217	.89	26.25	82.03	C	
Feb. 6.....	217	.89	26.50	82.81	C	

[a] Including 17 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=111.88$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.89$. $W=217$ grams.

$M=\frac{VPS}{W}=6{,}883$ kilograms per square centimeter (97,910 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 17 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, November 15, 1911.

	Grams.
Solid	2.4
Liquid (water)	65.5
Gaseous	140.9

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*° C.*	*Calories.*
Feb. 2	81.95	1.098	939.1
Do	81.95	1.102	942.6
Feb. 5	81.95	1.103	943.5

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 941.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 12	63.50	49.75	13.75
Do	63.50	50.00	13.50
Do	63.50	50.25	13.25

Average compression, 13.5 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 20	62	340	278	15
Do	62	332	270	15
Do	62	345	283	15

Average expansion of bore hole, 277 cubic centimeters (16.90 cubic inches).

GIANT COAL-MINE POWDER NO. 6.

Explosive, Giant coal-mine powder No. 6.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 201 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.14.
Color of explosive, corn.
Consistency, granular and fibrous, mostly very fine with a few large white crystals, wet, soft, moderately cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 12, 1911.

Unit swing on this date, 3.23 inches.

Weight of charge, in grams, 290, 290, 290.

Swing, in inches, 3.40, 3.37, 3.36.

Average swing, in inches, 3.38.

3.38 : 3.23 : : 290 : (277).

Therefore the unit deflective charge of Giant coal-mine powder No. 6 is 277 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Sept. 21	277	7.93	No ignition.	Oct. 5	277	.....	No ignition.
Do	277	7.97	Do.	Do	277	.....	Do.
Do	277	7.91	Do.	Do	277	.....	Do.
Sept. 23	277	8.27	Do.	Do	277	.....	Do.
Do	277	7.91	Do.	Do	277	.....	Do.
Do	277	8.01	Do.	Do	277	.....	Do.
Do	277	8.17	Do.	Do	277	.....	Do.
Sept. 25	277	8.27	Do.				
Do	277	8.14	Do.	TEST 4.			
Do	277	8.01	Do.				
				Sept. 18	[a] 680	4.29	No ignition.
TEST 3.				Sept. 19	[a] 680	4.22	Do.
				Do	[a] 680	4.22	Do.
Oct. 5	277	.....	No ignition.	Do	[a] 680	3.86	Do.
Do	277	.....	Do.	Sept. 20	[a] 680	3.99	Do.
Do	277	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 12	12.79	45.0	3,518
Sept. 18	12.99	45.0	3,464
Sept. 19	12.02	43.0	3,577

Average rate of detonation, 3,520 meters (11,550 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	13.25	16.74	5.00	0.250
Do	16.50	20.84	8.00	.400
Do	14.00	17.68	6.75	.338

Average height of flame, 18.42 inches.

Average duration of flame, 0.329 millisecond.

IMPACT TEST.

Date, February 3, 1913.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
70	1	Explosion.	25	1	No explosion.
60	1	Do.	28	1	Explosion.
50	1	Do.	26	1	No explosion.
40	1	Do.	27	1	Explosion.
30	1	Do.	26	1	Do.
20	1	No explosion.	25	4	No explosion.

The maximum height at which no explosion occurs established at 25 centimeters (9.84 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 183 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 29	3	Did not explode.	Feb. 29	2	Exploded.
Do	2	Do.	Do	3	Did not explode.
Do	1	Exploded.	Do	3	Do.
Do	2	Did not explode.			

The minimum distance at which no explosion occurs established at 3 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 8	215	1.14	24.50	76.56	A	
Do	215	1.14	23.50	73.44	A	74.48
Feb. 9	215	1.14	23.50	73.44	A	
Feb. 8	215	1.14	22.00	68.75	B	
Do	215	1.14	22.25	69.53	B	69.53
Do	215	1.14	22.50	70.31	B	
Feb. 6	215	1.14	20.00	62.50	C	
Feb. 7	215	1.14	20.50	64.06	C	63.28
Feb. 8	215	1.14	20.25	63.28	C	

[a] Including 15 grams of the original wrapper.

P=1.911A+0.5B−1.411C=87.81 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.14. W=215 grams.

$M=\frac{VPS}{W}$=6,984 kilograms per square centimeter (99,340 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 16, 1911.	Grams.
Solid	24.2
Liquid (water)	44.5
Gaseous	130.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 6	81.95	0.924	794.9
Do	81.95	.913	785.4
Do	81.95	.881	757.5

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 779.3.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 12	63.25	47.50	15.75
Do	63.25	48.25	15.00
Do	63.50	47.75	15.75

Average compression, 15.5 millimeters (0.61 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 20	62	250	188	15
Do	62	256	194	15
Do	62	259	197	15

Average expansion of bore hole, 193 cubic centimeters (11.77 cubic inches).

GIANT COAL-MINE POWDER NO. 7.

Explosive, Giant coal-mine powder No. 7.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 208 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.19.
Color of explosive, corn.
Consistency, granular and fibrous, mostly very fine with a few large white crystals, wet, soft, moderately cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 12, 1911.
Unit swing on this date, 3.23 inches.
Weight of charge, in grams, 245, 245, 245.
Swing, in inches, 3.27, 3.23, 3.19.
Average swing, in inches, 3.23.

3.23 : 3.23 : : 245 : (245).

Therefore the unit deflective charge of Giant coal-mine powder No. 7 is 245 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 25	245	8.33	No ignition.	Oct. 4	245	.....	No ignition.
Do	245	8.30	Do.	Do	245	.....	Do.
Do	245	8.20	Do.	Do	245	.....	Do.
Do	245	8.36	Do.	Do	245	.....	Do.
Do	245	8.35	Do.	Do	245	.....	Do.
Do	245	8.18	Do.	Do	245	.....	Do.
Do	245	8.26	Do.	Do	245	.....	Do.
Oct. 3	245	8.20	Do.	TEST 4.			
Do	245	8.44	Do.				
Do	245	7.98	Do.	Sept. 19	[a] 680	4.02	No ignition.
TEST 3.				Do	[a] 680	4.15	Do.
				Do	[a] 680	4.03	Do.
Oct. 4	245	.....	No ignition.	Sept. 20	[a] 680	4.06	Do.
Do	245	.....	Do.	Do	[a] 680	4.03	Do.
Do	245	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 20	13.30	45.0	3,383
Do	13.74	45.0	3,275
Do	13.87	45.0	3,244

Average rate of detonation, 3,301 meters (10,830 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	6.75	8.53	2.50	0.125
Do	12.25	15.47	4.50	.225
Do	15.50	19.58	8.00	.400

Average height of flame, 14.53 inches.
Average duration of flame, 0.250 millisecond.

IMPACT TEST.

Date, February 4, 1913.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	No explosion.	38	1	No explosion.
35	1	Do.	39	1	Explosion.
40	1	Explosion.	38	1	Do.
35	1	No explosion.	37	4	No explosion.
37	1	Do.			

The maximum height at which no explosion occurs established at 37 centimeters (14.57 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 191 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 29.....	3	Did not explode.	Feb. 29.....	3	Did not explode.
Do.....	2	Exploded.	Do.....	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 28.....	215.6	1.19	21.00	65.62	A	
Do.....	215.6	1.19	21.75	67.97	A	67.45
Do.....	215.6	1.19	22.00	68.75	A	
Feb. 23.....	215.6	1.19	19.50	60.94	B	
Do.....	215.6	1.19	19.00	59.38	B	59.90
Do.....	215.6	1.19	19.00	59.38	B	
Do.....	215.6	1.19	19.00	59.38	C	
Do.....	215.6	1.19	18.75	58.59	C	59.90
Do.....	215.6	1.19	19.75	61.72	C	

[a] Including 15.6 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=74.33$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.19$. $W=215.6$ grams.

$M=\frac{VPS}{W}=6{,}154$ kilograms per square centimeter (87,540 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 16 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 17, 1911.

	Grams.
Solid	33.3
Liquid (water)	48.5
Gaseous	126.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.8 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 7	81.95	0.991	851.6
Do	81.95	.997	856.8
Do	81.95	.978	840.2

[a] Including 7.8 grams of the original wrapper.

Average large calories per kilogram of explosive, 849.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 28	63.50	50.00	13.50
Do	63.00	49.50	13.50
Jan. 29	63.00	49.50	13.50

Average compression, 13.5 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 20	62	244	182	15
Do	62	248	186	15
Do	62	253	191	15

Average expansion of bore hole, 186 cubic centimeters (11.35 cubic inches).

GIANT COAL-MINE POWDER NO. 8.

Explosive, Giant coal-mine powder No. 8.
Class 2, hydrated.
Manufactured by the Giant Powder Co. (Consolidated).
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 231 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.29.
Color of explosive, corn.
Consistency, granular and fibrous, mostly very fine with a few large white crystals, wet, soft, moderately cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 12, 1911.
Unit swing on this date, 3.23 inches.
Weight of charge, in grams, 245, 245, 245.
Swing, in inches, 3.19, 3.24, 3.19.
Average swing, in inches, 3.21.

3.21 : 3.23 : : 245 : (247).

Therefore the unit deflective charge of Giant coal-mine powder No. 8 is 247 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Oct. 3	247	7.96	No ignition.	Oct. 4	247	.....	No ignition.
Do	247	8.18	Do.	Do	247	.....	Do.
Do	247	8.12	Do.	Do	247	.....	Do.
Do	247	8.39	Do.	Do	247	.....	Do.
Do	247	8.00	Do.	Do	247	.....	Do.
Do	247	8.01	Do.	Do	247	.....	Do.
Do	247	7.92	Do.	Do	247	.....	Do.
Do	247	8.09	Do.				
Do	247	8.23	Do.	TEST 4.			
Oct. 4	247	8.08	Do.				
				Sept. 20	a 680	4.13	No ignition.
TEST 3.				Do	a 680	4.24	Do.
				Do	a 680	4.20	Do.
Oct. 4	247	.....	No ignition.	Do	a 680	4.26	Do.
Do	247	.....	Do.	Do	a 680	3.98	Do.
Do	247	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 12	13.47	45.0	3,341
Sept. 19	13.40	45.0	3,358
Do	14.05	45.0	3,208

Average rate of detonation, 3,301 meters (10,830 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 9	17.25	21.79	7.50	0.375
Do	12.00	15.16	4.75	.238
Do	11.50	14.53	5.50	.225

Average height of flame, 17.16 inches.
Average duration of flame, 0.279 millisecond.

IMPACT TEST.

Date, February 4, 1913.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	No explosion.	36	3	No explosion.
35	1	Do.	36	1	Explosion.
40	1	Explosion.	35	1	Do.
38	1	Do.	34	5	No explosion.
37	1	Do.			

The maximum height at which no explosion occurs established at 34 centimeters (13.39 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 207 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 29	3	Did not explode.	Feb. 29	2	Did not explode.
Do	2	Do.	Do	2	Do.
Do	1	Exploded.			

The minimum distance at which no explosion occurs established at 2 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 10	214.2	1.29	24.75	77.34	A	76.04
Do	214.2	1.29	24.50	76.56	A	
Feb. 12	214.2	1.29	23.75	74.22	A	
Do	214.2	1.29	21.75	67.97	B	69.53
Do	214.2	1.29	22.75	71.09	B	
Do	214.2	1.29	22.25	69.53	B	
Do	214.2	1.29	22.75	71.09	C	69.53
Do	214.2	1.29	22.25	69.53	C	
Feb. 13	214.2	1.29	21.75	67.97	C	

[a] Including 14.2 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=81.97$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.29$. $W=214.2$ grams.

$M=\frac{VPS}{W}=7{,}405$ kilograms per square centimeter (105,330 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 21, 1911.	Grams.
Solid	31.4
Liquid (water)	48.5
Gaseous	119.0

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.1 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 8	81.95	0.959	829.2
Do	81.95	.985	851.9
Do	81.95	.983	850.1

[a] Including 7.1 grams of the original wrapper.

Average large calories per kilogram of explosive, 843.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 29	63.50	51.50	12.00
Do	63.25	50.75	12.50
Do	63.50	51.00	12.50

Average compression, 12.3 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 20	62	260	198	15
Do	62	266	204	15
Do	62	257	195	15

Average expansion of bore hole, 199 cubic centimeters (12.14 cubic inches)

GUARDIAN A.

Explosive, Guardian A.
Class 4, nitroglycerin.
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 176 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, September 30, 1912.

Unit swing on this date, 3.24 inches.

Weight of charge, in grams, 290, 290, 290.

Swing, in inches, 3.22, 3.13, 3.15.

Average swing, in inches, 3.17.

3.17 : 3.24 :: 290 : (296).

Therefore the unit deflective charge of Guardian A is 296 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 1	296	8.21	No ignition.	Oct. 3	296	.....	No ignition.
Oct. 2	296	8.34	Do.	Do	296	.....	Do.
Do	296	8.51	Do.	Do	296	.....	Do.
Do	296	8.25	Do.	Do	296	.....	Do.
Do	296	8.23	Do.	Do	296	.....	Do.
Do	296	8.20	Do.	Do	296	.....	Do.
Do	296	7.89	Do.	Do	296	.....	Do.
Do	296	8.50	Do.	TEST 4.			
Do	296	8.35	Do.				
Do	296	7.92	Do.	Oct. 2	[a] 680	4.31	No ignition.
TEST 3.				Oct. 3	[a] 680	4.11	Do.
				Do	[a] 680	4.10	Do.
Oct. 3	296	.....	No ignition.	Do	[a] 680	4.25	Do.
Do	296	.....	Do.	Do	[a] 680	4.12	Do.
Do	296	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 2	13.95	43.5	3,119
Do	14.20	43.5	3,063
Do	14.10	43.5	3,085

Average rate of detonation, 3,089 meters (10,130 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 20	11.00	13.89	4.25	0.212
Do	12.25	15.47	5.50	.275
Do	11.75	14.84	5.00	.250

Average height of flame, 14.73 inches.

Average duration of flame, 0.246 millisecond.

IMPACT TEST.

Date, October 18, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	31	1	Explosion.
25	1	Do.	30	2	No explosion.
35	1	Do.	30	1	Explosion.
45	1	Explosion.	28	1	Do.
40	1	Do.	26	1	Do.
37	1	Do.	25	1	Do.
36	1	Do.	20	1	Do.
35	1	Do.	18	1	No explosion.
30	1	No explosion.	19	5	Do.
33	1	Explosion.			

The maximum height at which no explosion occurs established at 19 centimeters (7.48 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 167 grams.

Date (1913).	Distance separating cartridges.	Result, upper cartridge.	Date (1913).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Mar. 20	7	Exploded.	Mar. 20	8	Did not explode.
Do	8	Did not explode.	Do	8	Do.

The minimum distance at which no explosion occurs established at 8 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 23	219	1.04	22.50	70.31	A	68.49
Do	219	1.04	21.75	67.97	A	
Do	219	1.04	21.50	67.19	A	
Oct. 24	219	1.04	20.50	64.06	B	64.58
Do	219	1.04	20.75	64.84	B	
Do	219	1.04	20.75	64.84	B	
Oct. 25	219	1.04	19.00	59.38	C	60.42
Do	219	1.04	19.75	61.72	C	
Do	219	1.04	19.25	60.16	C	

[a] Including 19 grams of the original wrapper.

P=1.911A+0.5B−1.411C=77.92 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.04. W=219 grams.

$M=\frac{VPS}{W}$=5,550 kilograms per square centimeter (78,940 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 19 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, October 3, 1912.

	Grams.
Solid	82.1
Liquid (water)	25.3
Gaseous	92.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Oct. 11	81.95	0.885	747.9
Oct. 14	81.95	.904	764.0
Oct. 16	81.95	.909	768.3

[a] Including 9.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 760.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Oct. 23	63.25	50.50	12.75
Do	63.50	51.25	12.25
Oct. 24	63.25	50.75	12.50

Average compression, 12.5 millimeters (0.49 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Oct. 17	62	212	150	15
Do	62	218	156	15
Do	62	219	157	15

Average expansion of bore hole, 154 cubic centimeters (9.39 cubic inches).

GUARDIAN NO. 2.

Explosive, Guardian No. 2.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 168 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.02.
Color of explosive, straw.
Consistency, powdered, very fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, December 4, 1911.
Unit swing on this date, 3.40 inches.
Weight of charge, in grams, 250, 250, 250.
Swing, in inches, 3.48, 3.52, 3.41.
Average swing, in inches, 3.47.

3.47 : 3.40 :: 250 : (245).

Therefore the unit deflective charge of Guardian No. 2 is 245 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Dec. 6	245	8.46	No ignition.	Dec. 14	245	.....	No ignition.
Dec. 7	245	8.05	Do.	Dec. 15	245	.....	Do.
Do	245	8.24	Do.	Do	245	.....	Do.
Do	245	8.08	Do.	Do	245	.....	Do.
Do	245	8.24	Do.	Do	245	.....	Do.
Do	245	8.35	Do.	Do	245	.....	Do.
Do	245	8.35	Do.	Do	245	.....	Do.
Do	245	8.40	Do.				
Do	245	8.34	Do.	TEST 4.			
Do	245	8.48	Do.	Dec. 11	[a] 680	3.81	No ignition.
TEST 3.				Do	[a] 680	4.19	Do.
				Do	[a] 680	4.20	Do.
Dec. 14	245	.....	No ignition.	Do	[a] 680	4.02	Do.
Do	245	.....	Do.	Do	[a] 680	4.15	Do.
Do	245	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911–12).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 30	16.10	44.5	2,764
Jan. 3	16.65	45.0	2,703
Do	16.20	44.5	2,747

Average rate of detonation, 2,738 meters (8,980 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 8	13.50	17.05	9.00	0.450
Do	14.75	18.63	5.75	.288
Do	15.75	19.89	6.00	.300

Average height of flame, 18.52 inches.
Average duration of flame, 0.346 millisecond.

IMPACT TEST.

Date, February 26, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
15	1	No explosion.	15	1	Explosion.
18	1	Explosion.	14	1	Do.
17	1	No explosion.	13	2	No explosion.
17	1	Explosion.	13	1	Explosion.
16	1	Do.	12	1	Do.
15	1	No explosion.	11	5	No explosion.

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 164 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 2	3	Did not explode.	Apr. 3	4	Exploded.
Do	2	Do.	Do	5	Do.
Do	1	Exploded.	Do	6	Did not explode.
Do	2	Do.	Do	6	Do.
Do	3	Did not explode.	Do	6	Do.
Do	3	Exploded.			

The minimum distance at which no explosion occurs established at 6 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 11	215	1.02	30.00	93.75	A	94.27
Do	215	1.02	30.75	96.09	A	
Do	215	1.02	29.75	92.97	A	
Mar. 20	215	1.02	28.75	89.84	B	90.10
Mar. 27	215	1.02	29.00	90.62	B	
Do	215	1.02	28.75	89.84	B	
Mar. 13	215	1.02	28.25	88.28	C	89.06
Mar. 12	215	1.02	28.75	89.84	C	
Mar. 20	215	1.02	28.50	89.06	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=99.54$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.02$. $W=215$ grams.

$M=\frac{VPS}{W}=7{,}084$ kilograms per square centimeter (100,760 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 8, 1911.	Grams.
Solid	33.1
Liquid (water)	63.0
Gaseous	106.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 21	81.95	1.228	1,060.1
Do	81.95	1.237	1,067.9
Do	81.95	1.221	1,054.0

[a] Including 7.5 grams of original wrapper.

Average large calories per kilogram of explosive, 1,060.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 28	63.50	49.75	13.75
Do	63.50	49.75	13.75
Do	63.50	50.00	13.50

Average compression, 13.7 millimeters (0.54 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 28	63	290	227	20
Do	63	290	227	20
Do	63	279	216	20

Average expansion of bore hole, 223 cubic centimeters (13.60 cubic inches)

GUARDIAN NO. 3.

Explosive, Guardian No. 3.
Class 1*a*, ammonium nitrate, containing nitroglycerin
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 150 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.92.
Color of explosive, straw.
Consistency, powdered, very fine, very dry, very soft, not cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, December 4, 1911.
Unit swing on this date, 3.40 inches.
Weight of charge, in grams, 270, 270, 270.
Swing, in inches, 3.36, 3.32, 3.45.
Average swing, in inches, 3.38.

3.38 : 3.40 : : 270 : (272).

Therefore the unit deflective charge of Guardian No. 3 is 272 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Dec. 7	272	8.49	No ignition.	Dec. 15	272	.....	No ignition.
Do	272	8.46	Do.	Do	272	.....	Do.
Do	272	8.30	Do.	Do	272	.....	Do.
Dec. 8	272	8.25	Do.	Do	272	.....	Do.
Do	272	7.92	Do.	Do	272	.....	Do.
Do	272	8.01	Do.	Do	272	.....	Do.
Do	272	8.06	Do.	Do	272	.....	Do.
Do	272	8.27	Do.	TEST 4.			
Do	272	8.23	Do.				
Do	272	8.29	Do.	Dec. 8	a 680	4.41	No ignition.
				Do	a 680	4.41	Do.
TEST 3.				Do	a 680	4.43	Do.
				Dec. 11	a 680	3.92	Do.
Dec. 15	272	.....	No ignition.	Dec. 12	a 680	4.13	Do.
Do	272	.....	Do.				
Do	272	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911–12).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 30	18.90	44.0	2,328
Jan. 3	19.35	44.5	2,300
Jan. 4	18.70	44.5	2,380

Average rate of detonation, 2,336 meters (7,660 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 8	14.25	18.00	4.50	0.225
Do	13.75	17.37	4.00	.200
Do	13.00	16.42	6.00	.300

Average height of flame, 17.26 inches.
Average duration of flame, 0.242 millisecond.

IMPACT TEST.

Date, February 23, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	22	1	Explosion.
25	1	Explosion.	21	1	Do.
24	1	Do.	20	1	Do.
23	1	Do.	15	1	Do.
22	1	No explosion.	14	5	No explosion.

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 148 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Apr. 15	5	Did not explode.	Apr. 23	4	Did not explode.
Apr. 17	4	Do.	Do	4	Exploded.
Do	3	Do.	Do	5	Did not explode.
Do	2	Exploded.	Do	5	Do.
Do	3	Do.			

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 29	219	0.92	26.25	82.03	A	82.29
Do	219	.92	26.00	81.25	A	
Do	219	.92	26.75	83.59	A	
Mar. 27	219	.92	24.50	76.56	B	78.39
Do	219	.92	25.50	79.69	B	
Do	219	.92	25.25	78.91	B	
Do	219	.92	26.25	82.03	C	82.29
Mar. 28	219	.92	26.25	82.03	C	
Do	219	.92	26.50	82.81	C	

[a] Including 19 grams of the original wrapper.

P=1.911A+0.5B−1.411C=80.34 kilograms per square centimeter.
V=15,000 cubic centimeters. S=0.92. W=219 grams.
$M=\frac{VPS}{W}$=5,063 kilograms per square centimeter (72,010 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 26 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 11, 1911.	Grams.
Solid	48.3
Liquid (water)	50.0
Gaseous	113.9

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 113.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 23	81.95	1.060	860.5
Feb. 26	81.95	1.086	890.9
Feb. 28	81.95	1.065	873.6

[a] Including 13 grams of the original wrapper.

Average large calories per kilogram of explosive, 878.0.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 29	63.00	51.25	11.75
Do	63.25	51.00	12.25
Do	63.25	51.50	11.75

Average compression, 11.9 millimeters (0.47 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 23	62	263	201	15
Do	62	266	204	15
Do	62	273	211	15

Average expansion of bore hole, 205 cubic centimeters (12.50 cubic inches).

GUARDIAN COAL POWDER B.

Explosive, Guardian coal powder B.
Class 4, nitroglycerin.
Manufactured by the Independent Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 213 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.28.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, May 6, 1912.
Unit swing on this date, 3.39 inches.
Weight of charge, in grams, 310, 310, 310.
Swing, in inches, 3.40, 3.38, 3.39.
Average swing, in inches, 3.39.

3.39 : 3.39 : : 310 : (310).

Therefore the unit deflective charge of Guardian coal powder B is 310 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
May 7	310	7.97	No ignition.	May 8	310	.....	No ignition
Do	310	8.01	Do.	Do	310	.....	Do.
Do	310	8.32	Do.	Do	310	.....	Do.
Do	310	8.22	Do.	Do	310	.....	Do.
Do	310	7.86	Do.	Do	310	.....	Do.
Do	310	8.06	Do.	Do	310	.....	Do.
Do	310	8.44	Do.	Do	310	.....	Do.
Do	310	8.18	Do.	TEST 4.			
Do	310	8.44	Do.	May 7	a 680	4.54	No ignition.
Do	310	8.12	Do.	Do	a 680	4.30	Do.
TEST 3.				Do	a 680	4.36	Do.
May 8	310	.....	No ignition.	May 8	a 680	4.34	Do.
Do	310	.....	Do.	Do	a 680	4.37	Do.
Do	310	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
May 7	16.50	44.0	2,667
Do	15.90	44.5	2,799
Do	16.10	44.0	2,733

Average rate of detonation, 2,733 meters (8,960 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 23	10.75	13.58	4.25	0.212
Do	10.50	13.26	4.00	.200
Do	6.75	8.53	2.50	.125

Average height of flame, 11.79 inches.
Average duration of flame, 0.179 millisecond.

Impact Test.

Date, August 20, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	14	1	No explosion.
30	1	Do.	16	1	Explosion.
20	1	Do.	15	5	No explosion.
10	1	No explosion.			

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 206 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 9	2	Did not explode.	Sept. 9	2	Did not explode.
Do	1	Exploded.	Do	2	Do.

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 26	215	1.28	21.50	67.19	A	
Do	215	1.28	21.25	66.41	A	67.19
Do	215	1.28	21.75	67.97	A	
Aug. 24	215	1.28	21.25	66.41	B	
Do	215	1.28	20.75	64.84	B	64.84
Do	215	1.28	20.25	63.28	B	
Aug. 23	215	1.28	20.50	64.06	C	
Do	215	1.28	20.50	64.06	C	63.54
Do	215	1.28	20.00	62.50	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=71.17$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.28$. $W=215$ grams.
$M=\frac{VPS}{W}=6{,}356$ kilograms per square centimeter (90,400 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 15 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, May 14, 1912.	Grams.
Solid	93.1
Liquid (water)	31.3
Gaseous	75.5

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 15	63.25	52.75	10.50
Do	63.25	53.25	10.00
Do	62.75	52.75	10.00

Average compression, 10.2 millimeters (0.40 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 27	62	168	106	15
Do	62	172	110	15
Do	62	169	107	15

Average expansion of bore hole, 108 cubic centimeters (6.59 cubic inches).

HECLA NO. 2.

Explosive, Hecla No. 2.
Class 1*a*, ammonium nitrate.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 252 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, straw.
Consistency, powdered, very fine, dry, hard, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 7, 1910.
Unit swing on this date, 2.99 inches.
Weight of charge, in grams, 240, 240, 240.
Swing, in inches, 2.81, 2.82, 2.89.
Average swing, in inches, 2.84.

2.84 : 2.99 : : 240 : (253).

Therefore the unit deflective charge of Hecla No. 2 is 253 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 11	253	8.33	No ignition.	Feb. 16	253	.....	No ignition.
Do	253	8.16	Do.	Do	253	.....	Do.
Do	253	8.26	Do.	Do	253	.....	Do.
Do	253	8.07	Do.	Do	253	.....	Do.
Do	253	8.07	Do.	Do	253	.....	Do.
Do	253	8.16	Do.	Do	253	.....	Do.
Do	253	8.07	Do.	Do	253	.....	Do.
Do	253	8.07	Do.				
Do	253	8.16	Do.	TEST 4.			
Feb. 12	253	8.07	Do.				
				Feb. 18	[a] 680	3.97	No ignition.
TEST 3.				Do	[a] 680	3.97	Do.
				Do	[a] 680	3.92	Do.
Feb. 16	253	.....	No ignition.	Do	[a] 680	3.87	Do.
Do	253	.....	Do.	Apr. 16	[a] 680	4.04	Do.
Do	253	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 4	10.34	43.0	4,159
Mar. 7	9.96	43.0	4,317
Do	9.96	43.0	4,317

Average rate of detonation, 4,264 meters (13,990 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 28	6.75	8.53	1.75	0.088
Do	3.00	3.79	1.00	.050
Feb. 21	7.00	8.84	1.75	.088

Average height of flame, 7.05 inches.
Average duration of flame, 0.075 millisecond.

IMPACT TEST.

Date, December 5, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
8	1	No explosion.	42	1	No explosion.
16	1	Do.	43	1	Do.
32	1	Do.	43	1	Explosion.
64	1	Explosion.	42	1	Do.
48	1	Do.	41	1	Do.
44	1	Do.	40	5	No explosion.

The maximum height at which no explosion occurs established at 40 centimeters (15.75 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 166 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 13	4	Did not explode.	Dec. 13	2	Did not explode.
Do	2	Do.	Do	2	Do.
Do	1	Exploded.			

The minimum distance at which no explosion occurs established at 2 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1912–13).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 22	217	1.03	22.00	91.67	A	89.24
Do	217	1.03	21.25	88.54	A	
Do	217	1.03	21.00	87.50	A	
Dec. 6	217	1.03	20.00	83.33	B	83.33
Feb. 13	217	1.03	20.25	84.38	B	
Do	217	1.03	19.75	82.29	B	
Jan. 22	217	1.03	19.00	79.17	C	80.56
Do	217	1.03	19.75	82.29	C	
Do	217	1.03	19.25	80.21	C	

[a] Including 17 grams of the original wrapper.

$P = 1.911A + 0.5B - 1.411C = 98.53$ kilograms per square centimeter.

$V = 15{,}000$ cubic centimeters. $S = 1.03$. $W = 217$ grams.

$M = \frac{VPS}{W} = 7{,}015$ kilograms per square centimeter (99,780 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 14, 1910.	Grams.
Solid	24.5
Liquid (water)	64.5
Gaseous	124.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 23	81.95	1.060	905.5
Do	81.95	1.094	934.8
Jan. 24	81.95	1.093	934.0

[a] Including 8.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 924.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Jan. 25	64.00	48.00	16.00
Do	63.75	47.25	16.50
Do	64.00	47.50	16.50

Average compression, 16.3 millimeters (0.64 inch).

EXPANSION OF BORE HOLE OF TRAUZL BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 15	63	310	247	15
Do	63	305	242	15
Nov. 16	63	316	253	15

Average expansion of bore hole, 247 cubic centimeters (15.07 cubic inches).

KANITE A.

Explosive, Kanite A.
Class 1*b*, ammonium nitrate.
Manufactured by the W. H. Blumenstein Chemical Works.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 289 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.91.
Color of explosive, corn.
Consistency, powdered, very fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, April 5, 1910.

Unit swing on this date, 3.44 inches.

Weight of charge, in grams, 260, 260, 260.

Swing, in inches, 3.54, 3.49, 3.50.

Average swing, in inches, 3.51.

3.51 : 3.44 : : 260 : (255).

Therefore the unit deflective charge of Kanite A is 255 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Apr. 6	255	7.99	No ignition.	Apr. 8	255	.....	No ignition.
Do	255	7.80	Do.	Do	255	.....	Do.
Do	255	7.90	Do.	Do	255	.....	Do.
Do	255	7.90	Do.	Do	255	.....	Do.
Do	255	7.90	Do.	Do	255	.....	Do.
Do	255	7.90	Do.	Do	255	.....	Do.
Do	255	7.80	Do.	Do	255	.....	Do.
Do	255	7.90	Do.	TEST 4.			
Do	255	7.80	Do.				
Apr. 7	255	7.80	Do.	Apr. 7	[a] 680	3.85	No ignition.
TEST 3.				Do	[a] 680	3.80	Do.
				Do	[a] 680	3.80	Do.
Apr. 8	255	.....	No ignition.	Do	[a] 680	3.81	Do.
Do	255	.....	Do.	Do	[a] 680	3.85	Do.
Do	255	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Apr. 6	13.06	43.0	3,292
Do	13.40	43.0	3,209
Do	12.99	43.0	3,310

Average rate of detonation, 3,270 meters (10,730 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Oct. 17	13.50	17.05	6.00	0.300
Do	14.00	17.68	5.50	.275
Do	16.25	20.52	7.75	.388

Average height of flame, 18.42 inches.

Average duration of flame, 0.321 millisecond.

IMPACT TEST.

Date, October 18, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
80	1	Explosion.	51	1	No explosion.
60	1	Do.	51	1	Explosion.
40	1	No explosion.	50	1	Do.
50	1	Do.	46	1	No explosion.
56	1	Explosion.	48	1	Explosion.
54	1	Do.	47	3	No explosion.
52	1	No explosion.	47	1	Explosion.
53	1	Explosion.	46	4	No explosion.
52	1	Do.			

The maximum height at which no explosion occurs established at 46 centimeters (18.11 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 143 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Oct. 31.....	2	Did not explode.	Oct. 31.....	1	Did not explode.
Do.....	1	Do.	Nov. 4.....	0	Exploded.
Do.....	1	Do.			

The minimum distance at which no explosion occurs is established at 1 inch.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 16.....	213	0.82	27.50	114.58	A	115.28
Do.....	213	.82	27.75	115.63	A	
Do.....	213	.82	27.75	115.63	A	
Apr. 15.....	213	.82	26.50	110.42	B	112.50
Do.....	213	.82	27.00	112.50	B	
Apr. 16.....	213	.82	27.50	114.58	B	
Do.....	213	.82	24.75	103.13	C	105.56
Do.....	213	.82	25.75	107.29	C	
Do.....	213	.82	25.50	106.25	C	

[a] Including 13 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=127.60$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.82$. $W=213$ grams.

$M=\frac{VPS}{W}=7{,}368$ kilograms per square centimeter (104,800 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, September 8, 1910.	Grams
Solid	12.5
Liquid (water)	86.0
Gaseous	110.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Apr. 17	81.95	1.138	988.4
Do	81.95	1.132	983.1
Do	81.95	1.155	1,003.3

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 991.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Oct. 23	63.25	49.00	14.25
Do	63.25	49.50	13.75
Do	62.50	48.75	13.75

Average compression, 13.9 millimeters (0.55 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 11	63	276	213	15
Do	63	270	207	15
Do	63	260	197	15

Average expansion of bore hole, 206 cubic centimeters (12.57 cubic inches).

LOMITE NO. 1.

Explosive, Lomite No. 1.
Class 2, hydrated.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 202 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.23.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by ballistic pendulum:

Date, October 28, 1912.
Unit swing on this date, 3.43 inches.
Weight of charge, in grams, 295, 295, 295.
Swing, in inches, 3.40, 3.37, 3.33.
Average swing, in inches, 3.37.

3.37 : 3.43 : : 295 : (300).

Therefore the unit deflective charge of Lomite No. 1 is 300 grams.

GAS AND DUST GALLERY No. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 31	300	8.18	No ignition.	Nov. 5	300	.....	No ignition.
Do	300	7.91	Do.	Do	300	.....	Do.
Do	300	7.86	Do.	Do	300	.....	Do.
Do	300	8.40	Do.	Do	300	.....	Do.
Do	300	8.53	Do.	Do	300	.....	Do.
Do	300	7.96	Do.	Do	300	.....	Do.
Do	300	8.23	Do.	Do	300	.....	Do.
Do	300	8.45	Do.	TEST 4.			
Do	300	8.26	Do.				
Do	300	8.53	Do.	Nov. 4	[a] 680	4.18	No ignition.
TEST 3.				Do	[a] 680	4.11	Do.
				Do	[a] 680	4.12	Do.
Nov. 5	300	.....	No ignition.	Do	[a] 680	4.18	Do.
Do	300	.....	Do.	Do	[a] 680	4.11	Do.
Do	300	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 31	9.90	42.0	4,242
Nov. 1	9.75	42.0	4,308
Nov. 2	10.00	42.0	4,200

Average rate of detonation, 4,250 meters (13,940 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Feb. 20	10.25	12.95	6.75	0.338
Do	8.75	11.05	5.25	.262
Do	9.00	11.37	6.75	.338

Average height of flame, 11.79 inches.
Average duration of flame, 0.313 millisecond.

IMPACT TEST.

Date, November 8, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
10	1	No explosion.	14	1	No explosion.
20	1	Explosion.	15	5	Do.
16	1	Do.			

The maximum height at which no explosion occurs established at 15 centimeters (5.91 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 198 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 11.....	10	Did not explode.	Nov. 11.....	9	Did not explode.
Do.....	6	Exploded.	Do.....	9	Do.
Do.....	8	Do.	Do.....	9	Do.

The minimum distance at which no explosion occurs established at 9 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Dec. 2.....	212	1.23	15.75	65.63	A	
Do.....	212	1.23	15.50	64.58	A	65.28
Do.....	212	1.23	15.75	65.63	A	
Do.....	212	1.23	14.25	59.38	B	
Do.....	212	1.23	14.50	60.42	B	60.07
Do.....	212	1.23	14.50	60.42	B	
Do.....	212	1.23	14.75	61.46	C	
Do.....	212	1.23	14.75	61.46	C	61.46
Do.....	212	1.23	14.75	61.46	C	

[a] Including 12 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=68.07$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.23$. $W=212$ grams.

$M=\frac{VPS}{W}=5{,}924$ kilograms per square centimeter (84,260 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 12 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 6, 1912.	Grams.
Solid	69.1
Liquid (water)	18.7
Gaseous	114.6

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 22	81.95	0.820	715.2
Do	81.95	.784	683.4
Do	81.95	.781	680.8

[a] Including 6 grams of the original wrapper.

Average large calories per kilogram of explosive, 693.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1913).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Feb. 25	63.50	47.00	16.50
Do	63.50	47.25	16.25
Do	63.50	47.50	16.00

Average compression, 16.2 millimeters (0.64 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 14	62	228	166	15
Do	62	221	159	15
Do	62	225	163	15

Average expansion of bore hole, 163 cubic centimeters (9.94 cubic inches).

MINE-ITE A.

Explosive, Mine-ite A.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 203 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.14.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, August 6, 1909.
Unit swing on this date, 2.79 inches.
Weight of charge, 315, 315, 315.
Swing, in inches, 2.95, 2.98, 2.97.
Average swing, in inches, 2.97.

2.97 : 2.79 : : 315 : (296).

Therefore the unit deflective charge of Mine-ite A is 296 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 7	296	8.30	No ignition.	Aug. 9	296	.....	No ignition.
Do	296	8.30	Do.	Do	296	.....	Do.
Do	296	8.47	Do.	Do	296	.....	Do.
Do	296	8.30	Do.	Do	296	.....	Do.
Do	296	8.47	Do.	Do	296	.....	Do.
Do	296	8.47	Do.	Do	296	.....	Do.
Aug. 9	296	8.23	Do.	Do	296	.....	Do.
Do	296	8.39	Do.				
Do	296	8.23	Do.	TEST 4.			
Aug. 10	296	8.17	Do.				
				Aug. 6	[a] 680	4.23	No ignition.
TEST 3.				Do	[a] 680	4.40	Do.
				Do	[a] 680	4.40	Do.
Aug. 9	296	.....	No ignition.	Do	[a] 680	4.40	Do.
Do	296	.....	Do.	Do	[a] 680	4.02	Do.
Do	296	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 3	12.57	43.0	3,421
Do	12.36	43.0	3,479
Do	12.54	43.0	3,429

Average rate of detonation, 3,443 meters (11,290 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 24	14.50	18.32	6.25	0.312
Do	13.25	16.74	6.25	.312
Do	16.00	20.21	7.50	.375

Average height of flame, 18.42 inches.
Average duration of flame, 0.333 millisecond.

Impact Test.

Date, May 3, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	12	1	Explosion.
12	1	No explosion.	10	1	Do.
16	1	Explosion.	9	2	No explosion.
13	1	No explosion.	9	1	Explosion.
14	1	Explosion.	8	5	No explosion.
13	1	Do.			

The maximum height at which no explosion occurs established at 8 centimeters (3.15 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 182 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 5	10	Exploded.	Dec. 6	16	Did not explode.
Do	13	Did not explode.	Do	16	Exploded.
Do	12	Exploded.	Do	17	Do.
Do	14	Did not explode.	Do	18	Do.
Do	13	Do.	Do	19	Did not explode.
Do	13	Exploded.	Do	19	Exploded.
Dec. 6	14	Do.	Dec. 7	20	Did not explode.
Do	15	Did not explode.	Do	20	Do.
Do	15	Do.	Do	20	Do.
Do	15	Exploded.			

The minimum distance at which no explosion occurs established at 20 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 23	[b] 213	1.13	14.75	61.46	A	
Do	213	1.13	20.00	62.50	A	62.41
Do	213	1.13	20.25	63.28	A	
Do	213	1.13	19.75	61.72	B	
Do	213	1.13	20.00	62.50	B	62.50
Do	213	1.13	20.25	63.28	B	
Do	213	1.13	19.75	61.72	C	
Do	213	1.13	19.75	61.72	C	61.20
Jan. 24	213	1.13	19.25	60.16	C	

[a] Including 13 grams of the original wrapper.
[b] Indicator spring, 0.24 millimeter=1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=64.16$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.13$. $W=213$ grams.

$M=\frac{VPS}{W}=5{,}106$ kilograms per square centimeter (72,620 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, February 27, 1913.	Grams.
Solid	66.1
Liquid (water)	20.0
Gaseous	107.9

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams. [a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 4	81.95	0.822	713.6
Jan. 6	81.95	.792	687.2
Do	81.95	.811	703.9

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 701.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 17	63.50	50.00	13.50
Do	63.50	50.00	13.50
Do	63.50	50.00	13.50

Average compression, 13.5 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 19	62	254	192	15
Do	62	253	191	15
Do	62	262	200	15

Average expansion of bore hole, 194 cubic centimeters (11.83 cubic inches).

MINE-ITE A-2.

Explosive, Mine-ite A–2.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 151 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, March 11, 1912.
Unit swing on this date, 3.50 inches.
Weight of charge, in grams, 310, 310, 310.
Swing, in inches, 3.32, 3.46, 3.42.
Average swing, in inches, 3.40.

3.40 : 3.50 : : 310 : (319).

Therefore the unit deflective charge of Mine-ite A–2 is 319 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Mar. 12	319	7.99	No ignition.	Mar. 15	319	.....	No ignition.
Do	319	8.23	Do.	Do	319	.....	Do.
Do	319	8.09	Do.	Do	319	.....	Do.
Mar. 13	319	7.76	Do.	Do	319	.....	Do.
Do	319	7.87	Do.	Do	319	.....	Do.
Do	319	7.98	Do.	Do	319	.....	Do.
Do	319	7.91	Do.	Do	319	.....	Do.
Do	319	7.90	Do.				
Do	319	7.88	Do.	TEST 4.			
Do	319	7.85	Do.				
				Mar. 14	[a] 680	4.05	No ignition.
TEST 3.				Do	[a] 680	4.03	Do.
				Do	[a] 680	4.21	Do.
Mar. 14	319	.....	No ignition.	Do	[a] 680	4.14	Do.
Do	319	.....	Do.	Do	[a] 680	4.08	Do.
Mar. 15	319	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 11	13.85	45.0	3,249
Mar. 15	13.70	44.5	3,243
Do	13.65	44.5	3,260

Average rate of detonation, 3,252 meters (10,670 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
May 14	11.75	14.84	4.00	0.200
Do	9.25	11.68	3.00	.150
Do	12.00	15.16	4.00	.200

Average height of flame, 13.89 inches.
Average duration of flame, 0.183 millisecond.

Impact Test.

Date, April 20, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	18	1	Explosion.
18	1	No explosion.	17	1	Do.
19	3	Do.	16	5	No explosion.
19	1	Explosion.			

The maximum height at which no explosion occurs established at 16 centimeters (6.30 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 150 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 3	2	Exploded.	May 3	10	Did not explode.
Do	4	Do.	Do	9	Do.
Do	6	Do.	Do	9	Do.
Do	8	Do.	Do	9	Do.

The minimum distance at which no explosion occurs established at 9 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
May 9	216	0.93	20.00	62.50	A	
Do	216	.93	19.25	60.16	A	61.72
Do	216	.93	20.00	62.50	A	
May 8	216	.93	20.25	63.28	B	
Do	216	.93	19.50	60.94	B	61.72
Do	216	.93	19.50	60.94	B	
Do	216	.93	19.00	59.38	C	
May 9	216	.93	18.25	57.03	C	58.60
Do	216	.93	19.00	59.38	C	

[a] Including 16 grams of the original wrapper.

P=1.911A+0.5B−1.411C=66.12 kilograms per square centimeter.
V=15,000 cubic centimeters. S=0.93. W=216 grams.

$M=\frac{VPS}{W}$=4,270 kilograms per square centimeter (60,730 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 16 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, March 8, 1912.	Grams.
Solid	58.5
Liquid (water)	15.5
Gaseous	129.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
May 12	81.95	0.790	679.2
Do	81.95	.792	680.8
Do	81.95	.826	710.4

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 690.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 18	63.25	50.50	12.75
Do	63.25	50.75	12.50
Do	63.25	50.50	12.75

Average compression, 12.7 millimeters (0.50 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 30	62	211	149	16
Do	62	218	156	16
Do	62	212	150	16

Average expansion of bore hole, 152 cubic centimeters (9.27 cubic inches).

MINE-ITE B.

Explosive, Mine-ite B.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 183 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, straw.
Consistency, granular and fibrous,, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, August 4, 1909.
Unit swing on this date, 2.95 inches.
Weight of charge, in grams, 340, 340, 340.
Swing, in inches, 2.95, 2.92, 2.97.
Average swing, in inches, 2.95.

2.95 : 2.95 : : 340 : (340).

Therefore the unit deflective charge of Mine-ite B is 340 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 4	340	7.88	No ignition.	Aug. 5	340	.....	No ignition.
Do	340	8.63	Do.	Do	340	.....	Do.
Do	340	8.13	Do.	Do	340	.....	Do.
Do	340	8.13	Do.	Do	340	.....	Do.
Do	340	8.37	Do.	Do	340	.....	Do.
Do	340	8.37	Do.	Do	340	.....	Do.
Aug. 5	340	8.30	Do.	Do	340	.....	Do.
Do	340	8.47	Do.				
Do	340	8.50	Do.	TEST 4.			
Do	340	8.47	Do.				
				Aug. 2	[a] 680	4.14	No ignition.
TEST 3.				Do	[a] 680	3.98	Do.
				Do	[a] 680	3.81	Do.
Aug. 5	340	.....	No ignition.	Do	[a] 680	3.90	Do.
Do	340	.....	Do.	Do	[a] 680	3.81	Do.
Do	340	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909–10).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 25	14.75	43.0	2,915
June 3	15.00	43.0	2,867
Do	14.63	43.0	2,941

Average rate of detonation, 2,908 meters (9,540 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 24	13.00	16.42	5.50	0.275
Do	12.25	15.47	6.25	.312
Do	9.50	12.00	5.00	.250

Average height of flame, 14.63 inches.

Average duration of flame, 0.279 millisecond.

Impact Test.

Date, May 5, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
16	1	Explosion.	9	1	Explosion.
10	1	Do.	8	2	No explosion.
6	1	No explosion.	8	1	Explosion.
8	1	Do.	7	5	No explosion.

The maximum height at which no explosion occurs established at 7 centimeters (2.76 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 159 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 12	6	Did not explode.	Dec. 12	5	Exploded.
Do	4	Exploded.	Do	6	Did not explode.
Do	5	Did not explode.	Do	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 25	214	0.99	18.75	58.59	A	
Do	214	.99	18.75	58.59	A	58.33
Do	214	.99	18.50	57.81	A	
Jan. 24	214	.99	17.25	53.91	B	
Jan. 25	214	.99	17.50	54.69	B	54.17
Do	214	.99	17.25	53.91	B	
Jan. 24	214	.99	17.25	53.91	C	
Do	214	.99	17.25	53.91	C	53.65
Do	214	.99	17.00	53.12	C	

[a] Including 14 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=62.85$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.99$. $W=214$ grams.

$M=\frac{VPS}{W}=4{,}361$ kilograms per square centimeter (62,030 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 11, 1912.	Grams.
Solid	77.7
Liquid (water)	15.5
Gaseous	103.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 17	81.95	0.773	667.5
Do	81.95	.806	696.3
Do	81.95	.789	681.4

[a] Including 7 grams of the original wrapper.

Average large calories per kilogram of explosive, 681.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 12	63.75	53.25	10.50
Do	63.75	53.25	10.50
Do	63.75	53.25	10.50

Average compression, 10.5 millimeters (0.41 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Mar. 19	62	220	158	15
Do	62	222	160	15
Do	62	221	159	15

Average expansion of bore hole, 159 cubic centimeters (9.70 cubic inches).

MINE-ITE B-2.

Explosive, Mine-ite B-2.
Class 4, nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 127 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.77.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by ballistic pendulum:
Date, March 11, 1912.
Unit swing on this date, 3.50 inches.
Weight of charge, in grams, 325, 325, 325.
Swing, in inches, 3.38, 3.42, 3.43.
Average swing, in inches, 3.41.

$$3.41 : 3.50 :: 325 : (334).$$

Therefore the unit deflective charge of Mine-ite B-2 is 334 grams.

Gas and Dust Gallery No. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
Test 1.	*Grams.*	*Per cent.*	
Mar. 12	334	8.04	No ignition.
Do	334	7.96	Do.
Do	334	8.16	Do.
Do	334	8.09	Do.
Do	334	7.94	Do.
Do	334	7.83	Do.
Do	334	7.91	Do.
Do	334	8.06	Do.
Do	334	8.24	Do.
Do	334	8.08	Do.
Test 3.			
Mar. 14	334	.....	No ignition.
Do	334	.....	Do.
Do	334	.....	Do.
Test 3—Con.	*Grams.*	*Per cent.*	
Mar. 14	334	.....	No ignition.
Do	334	.....	Do.
Do	334	.....	Do.
Do	334	.....	Do.
Do	334	.....	Do.
Do	334	.....	Do.
Do	334	.....	Do.
Test 4.			
Mar. 13	[a] 680	3.97	No ignition.
Do	[a] 680	3.95	Do.
Do	[a] 680	4.18	Do.
Do	[a] 680	4.12	Do.
Mar. 14	[a] 680	3.79	Do.

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 11	20.00	44.5	2,225
Mar. 16	20.00	45.0	2,250
Do	19.30	44.5	2,306

Average rate of detonation, 2,260 meters (7,410 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
May 13	8.00	10.11	3.00	0.150
Do	9.00	11.37	3.50	.175
Do	8.75	11.05	3.50	.175

Average height of flame, 10.84 inches.
Average duration of flame, 0.167 millisecond.

Impact Test.

Date, April 19, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	25	1	Explosion.
35	1	Do.	23	1	Do.
30	1	Do.	21	1	Do.
25	1	No explosion.	19	1	No explosion.
28	1	Explosion.	20	5	Do.
26	1	Do.			

The maximum height at which no explosion occurs established at 20 centimeters (7.87 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 124 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 2	3	Did not explode.	May 2	2	Did not explode.
Do	1	Do.	May 3	2	Do.
Do	0	Exploded.	Do	2	Do.
Do	1	Do.			

The minimum distance at which no explosion occurs established at 2 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter = 1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
May 9	217	0.77	17.25	53.91	A	53.13
Do	217	.77	16.50	51.56	A	
Do	217	.77	17.25	53.91	A	
May 7	217	.77	17.00	53.12	B	52.60
May 8	217	.77	16.50	51.56	B	
Do	217	.77	17.00	53.12	B	
May 7	217	.77	16.00	50.00	C	51.56
Do	217	.77	16.75	52.34	C	
Do	217	.77	16.75	52.34	C	

[a] Including 17 grams of the original wrapper.

P=1.911A+0.5B−1.411C=55.08 kilograms per square centimeter.
V=15,000 cubic centimeters. S=0.77. W=217 grams.
$M=\frac{VPS}{W}=2{,}932$ kilograms per square centimeter (41,700 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 17 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, March 7, 1912.	Grams.
Solid	67.4
Liquid (water)	12.6
Gaseous	126.2

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 18	63.25	55.75	7.50
Do	63.25	55.75	7.50
Do	63.25	55.75	7.50

Average compression, 7.5 millimeters (0.30 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	°*C.*
Apr. 29	62	218	156	16
Apr. 30	62	220	158	16
Do	62	214	152	16

Average expansion of bore hole, 155 cubic centimeters (9.46 cubic inches).

MINE-ITE NO. 5–D.

Explosive, Mine-ite No. 5–D.
Class, 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the Burton Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 150 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.90.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, August 29, 1912.

Unit swing on this date, 3.44 inches.

Weight of charge, in grams, 240, 240, 240.

Swing, in inches, 3.49, 3.33, 3.48.

Average swing, in inches, 3.43.

3.43 : 3.44 : : 240 : (241).

Therefore the unit deflective charge of Mine-ite No. 5–D is 241 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 29	241	7.87	No ignition.	Sept. 3	241	.....	No ignition.
Do	241	8.25	Do.	Do	241	.....	Do.
Do	241	7.85	Do.	Do	241	.....	Do.
Do	241	7.93	Do.	Do	241	.....	Do.
Aug. 30	241	8.02	Do.	Do	241	.....	Do.
Do	241	7.93	Do.	Do	241	.....	Do.
Do	241	8.01	Do.	Do	241	.....	Do.
Do	241	7.80	Do.				
Do	241	8.24	Do.	TEST 4.			
Do	241	8.30	Do.				
				Aug. 31	[a] 680	4.21	No ignition.
TEST 3.				Do	[a] 680	4.04	Do.
				Do	[a] 680	4.01	Do.
Sept. 3	241	.....	No ignition.	Do	[a] 680	4.14	Do.
Do	241	.....	Do.	Do	[a] 680	4.22	Do.
Do	241	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 30	16.45	43.0	2,614
Do	15.80	43.0	2,722
Do	16.30	43.0	2,638

Average rate of detonation, 2,658 meters (8,720 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 9	8.00	10.11	3.75	0.188
Sept. 10	4.25	5.37	1.50	.075
Do	6.25	7.89	1.50	.075

Average height of flame, 7.79 inches.

Average duration of flame, 0.113 millisecond.

IMPACT TEST.

Date, September 27, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	Explosion.	13	1	Explosion.
20	1	Do.	11	1	Do.
15	1	Do.	10	5	No explosion.

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 145 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 3	3	Did not explode.	Sept. 3	2	Exploded.
Do	2	Do.	Do	3	Did not explode.
Do	1	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 18	218	0.90	30.00	93.75	A	
Oct. 21	218	.90	29.50	92.19	A	92.19
Do	218	.90	29.00	90.62	A	
Oct. 18	218	.90	28.00	87.50	B	
Do	218	.90	28.50	89.06	B	88.28
Do	218	.90	28.25	88.28	B	
Oct. 7	218	.90	26.75	83.59	C	
Oct. 9	218	.90	27.25	85.16	C	83.59
Do	218	.90	26.25	82.03	C	

[a] Including 18 grams of the original wrapper.

P=1.911A+0.5B−1.411C=102.37 kilograms per square centimeter.

V=15,000 cubic centimeters. S=0.90. W=218 grams.

$M=\frac{VPS}{W}$=6,339 kilograms per square centimeter (90,160 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, September 4, 1912.

	Grams.
Solid	20.5
Liquid (water)	68.0
Gaseous	119.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Sept. 14	81.95	1.155	982.8
Sept. 16	81.95	1.169	994.9
Do	81.95	1.115	948.5

[a] Including 9 grams of the original wrapper.

Average large calories per kilogram of explosive, 975.4.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 23	63.25	50.75	12.50
Do	63.25	51.25	12.00
Sept. 24	63.25	51.25	12.00

Average compression, 12.2 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Oct. 17	62	291	229	15
Do	62	291	229	15
Do	62	294	232	15

Average expansion of bore hole, 230 cubic centimeters (14.03 cubic inches).

MONOBEL NO. 2.

Explosive, Monobel No. 2.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 236 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.98.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 7, 1910.

Unit swing on this date, 2.99 inches.

Weight of charge, in grams, 240, 240, 240.

Swing, in inches, 2.95, 3.04, 3.02.

Average swing, in inches, 3.00.

3.00 : 2.99 : : 240 : (239).

Therefore the unit deflective charge of Monobel No. 2 is 239 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 10	239	8.26	No ignition.	Feb. 15	239	.....	No ignition.
Do	239	7.97	Do.	Do	239	.....	Do.
Do	239	7.92	Do.	Do	239	.....	Do.
Do	239	7.97	Do.	Do	239	.....	Do.
Do	239	8.16	Do.	Feb. 16	239	.....	Do.
Do	239	7.97	Do.	Do	239	.....	Do.
Do	239	7.87	Do.	Do	239	.....	Do.
Do	239	8.06	Do.				
Feb. 11	239	7.87	Do.	TEST 4.			
Do	239	8.16	Do.				
				Feb. 18	[a] 680	3.83	No ignition.
TEST 3.				Do	[a] 680	4.02	Do.
				Do	[a] 680	3.92	Do.
Feb. 15	239	.....	No ignition.	Do	[a] 680	4.02	Do.
Do	239	.....	Do.	Do	[a] 680	3.87	Do.
Do	239	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 7	14.34	43.0	2,999
Feb. 8	14.30	43.0	3,007
Do	14.23	43.0	3,022

Average rate of detonation, 3,009 meters (9,870 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 29	6.00	7.58	1.50	0.075
Do	7.75	9.79	2.00	.100
Do	8.50	10.74	2.00	.100

Average height of flame, 9.37 inches.

Average duration of flame, 0.092 millisecond.

IMPACT TEST.

Date, March 2, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
30	1	Explosion.	21	1	Explosion.
24	1	No explosion.	20	1	No explosion.
26	1	Explosion.	20	1	Explosion.
24	1	Do.	19	1	Do.
23	1	No explosion.	18	1	Do.
23	1	Explosion.	17	1	No explosion.
22	1	No explosion.	17	1	Explosion.
22	1	Explosion.	16	5	No explosion.

The maximum height at which no explosion occurs established at 16 centimeters (6.30 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 162 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 13	6	Exploded.	Dec. 13	10	Exploded.
Do	10	Did not explode.	Do	11	Did not explode.
Do	8	Exploded.	Do	11	Do.
Do	9	Do.	Do	11	Do.

The minimum distance at which no explosion occurs established at 11 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 29	[b] 218	0.98	37.50	93.75	A	92.97
Do	218	.98	29.50	92.19	A	
Oct. 31	218	.98	29.75	92.97	A	
Nov. 25	218	.98	28.50	89.06	B	88.54
Do	218	.98	28.50	89.06	B	
Nov. 28	218	.98	28.00	87.50	B	
Do	218	.98	27.25	85.16	C	84.38
Do	218	.98	26.25	82.03	C	
Do	218	.98	27.50	85.94	C	

[a] Including 18 grams of the original wrapper.
[b] Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=102.88$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.98$. $W=218$ grams.

$M=\frac{VPS}{W}=6{,}937$ kilograms per square centimeter (98,670 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 19 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 8, 1910.

	Grams.
Solid	20.4
Liquid (water)	67.5
Gaseous	128.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.0 grams.[a]

Date (1910).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Nov. 16	79.85	0.968	803.9
Nov. 17	79.85	.945	784.5
Nov. 18	79.85	.950	788.7

[a] Including 9 grams of the original wrapper.

Average large calories per kilogram of explosive, 792.4

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 15	63.25	49.00	14.25
Do	63.25	49.50	13.75
Do	63.25	49.00	14.25

Average compression, 14.1 millimeters (0.56 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 11	63	316	253	15
Do	63	317	254	15
Do	63	326	263	15

Average expansion of bore hole, 257 cubic centimeters (15.68 cubic inches).

MONOBEL NO. 3.

Explosive, Monobel No. 3.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 221 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, straw.
Consistency, granular and fibrous, fine, dry, hard, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 14, 1910.

Unit swing on this date, 3.37 inches.

Weight of charge, in grams, 250, 250, 250.

Swing, in inches, 3.50, 3.41, 3.49.

Average swing, in inches, 3.47.

3.47 : 3.37 : : 250 : (243).

Therefore the unit deflective charge of Monobel No. 3 is 243 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 14	243	8.05	No ignition.	Feb. 17	243	.....	No ignition.
Do	243	8.13	Do.	Do	243	.....	Do.
Feb. 15	243	8.13	Do.	Do	243	.....	Do.
Do	243	8.13	Do.	Do	243	.....	Do.
Do	243	8.05	Do.	Do	243	.....	Do.
Do	243	7.77	Do.	Do	243	.....	Do.
Do	243	8.13	Do.	Do	243	.....	Do.
Do	243	7.95	Do.				
Do	243	8.13	Do.	TEST 4.			
Do	243	8.13	Do.				
				Mar. 3	a 680	3.87	No ignition.
TEST 3.				Mar. 7	a 680	3.92	Do.
				Do	a 680	3.97	Do.
Feb. 17	243	.....	No ignition.	Do	a 680	3.87	Do.
Do	243	.....	Do.	Do	a 680	3.87	Do.
Do	243	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Mar. 7	19.58	43.0	2,196
Mar. 8	19.37	43.0	2,220
Do	19.36	43.0	2,221

Average rate of detonation, 2,212 meters (7,260 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1913).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Jan. 29	6.75	8.53	1.75	0.088
Do	8.75	11.05	2.50	.125
Do	7.75	9.79	2.25	.112

Average height of flame, 9.79 inches.

Average duration of flame, 0.108 millisecond.

IMPACT TEST.

Date, March 2, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	18	3	No explosion.
20	1	Explosion.	18	1	Explosion.
19	1	No explosion.	17	1	Do.
19	1	Explosion.	16	5	No explosion.

The maximum height at which no explosion occurs is established at 16 centimeters (6.30 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 150 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 29	4	Did not explode.	Nov. 30	5	Did not explode.
Do	3	Exploded.	Do	5	Do.
Do	4	Did not explode.	Do	5	Do.
Do	4	Exploded.			

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Oct. 19	220	0.93	37.00	92.50	A	
Oct. 20	220	.93	36.50	91.25	A	91.25
Do	220	.93	36.00	90.00	A	
Oct. 21	220	.93	34.00	85.00	B	
Do	220	.93	34.50	86.25	B	85.83
Do	220	.93	34.50	86.25	B	
Oct. 20	220	.93	32.50	81.25	C	
Do	220	.93	31.50	78.75	C	79.79
Do	220	.93	31.75	79.37	C	

[a] Including 20 grams of the original wrapper.

P=1.911A+0.5B−1.411C=104.71 kilograms per square centimeter.

V=15,000 cubic centimeters. S=0.93. W=220 grams.

$M=\frac{VPS}{W}$ =6,640 kilograms per square centimeter (94,450 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 20 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 11, 1910.

	Grams.
Solid	23.0
Liquid (water)	67.0
Gaseous	130.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1910).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Nov. 18	80.05	0.919	757.4
Do	80.05	.911	750.8
Nov. 19	80.05	.915	754.1

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 754.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 15	63.25	53.25	10.00
Do	63.25	53.25	10.00
Do	63.25	54.00	9.25

Average compression, 9.8 millimeters (0.39 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Nov. 11	63	310	247	15
Do	63	310	247	15
Do	63	319	256	15

Average expansion of bore hole, 250 cubic centimeters (15.25 cubic inches).

MONOBEL NO. 4.

Explosive, Monobel No. 4.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, $\frac{7}{8}$ inch.
Length of cartridge, 8 inches.
Average weight, 95 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.08.
Color of explosive, corn.
Consistency, fibrous, fine, very dry, very soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, January 23, 1911.
Unit swing on this date, 3.05 inches.
Weight of charge, in grams, 250, 250, 250.
Swing, in inches, 3.05, 3.13, 3.02.
Average swing, in inches, 3.07.

3.07 : 3.05 : : 250 : (248).

Therefore the unit deflective charge of Monobel No. 4 is 248 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Jan. 26	248	8.02	No ignition.	Feb. 3	248	.....	No ignition.
Do	248	8.20	Do.	Do	248	.....	Do.
Do	248	8.10	Do.	Do	248	.....	Do.
Do	248	8.03	Do.	Do	248	.....	Do.
Do	248	8.04	Do.	Feb. 4	248	.....	Do.
Jan. 27	248	7.94	Do.	Do	248	.....	Do.
Do	248	7.94	Do.	Do	248	.....	Do.
Do	248	7.96	Do.				
Do	248	7.89	Do.	TEST 4.			
Do	248	8.02	Do.				
				Jan. 31	a 680	3.98	No ignition.
TEST 3.				Do	a 680	3.93	Do.
				Feb. 1	a 680	4.07	Do.
Feb. 3	248	.....	No ignition.	Do	a 680	4.02	Do.
Do	248	.....	Do.	Do	a 680	4.12	Do.
Do	248	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 31	16.37	45.0	2,749
Do	16.50	45.0	2,727
Do	16.43	45.0	2,739

Average rate of detonation, 2,738 meters (8,980 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Aug. 14	10.00	12.63	2.50	0.125
Sept. 28	8.75	11.05	3.00	.150
Do	9.50	11.93	2.50	.125

Average height of flame, 11.87 inches.
Average duration of flame, 0.133 millisecond.

IMPACT TEST.

Date, July 26, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	13	1	Explosion.
18	1	Do.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 184 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 23	3	Exploded.	Sept. 27	4	Did not explode.
Do	5	Did not explode.	Do	4	Do.
Sept. 25	4	Do.			

The minimum distance at which no explosion occurs established at 4 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
July 26	[b] 222	1.08	35.00	87.50	A	
Do	222	1.08	28.75	89.85	A	88.54
Do	222	1.08	28.25	88.28	A	
July 27	222	1.08	25.00	78.12	B	
Do	222	1.08	25.50	79.69	B	79.69
Do	222	1.08	26.00	81.25	B	
Do	222	1.08	22.25	69.53	C	
Do	222	1.08	23.25	72.66	C	71.09
Do	222	1.08	22.75	71.09	C	

[a] Including 22 grams of the original wrapper.

[b] Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

$P=1.911A+0.5B-1.411C=108.74$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.08$. $W=222$ grams.

$M=\frac{VPS}{W}=7{,}935$ kilograms per square centimeter (112,870 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 22 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 19, 1911.	Grams.
Solid	22.7
Liquid (water)	63.5
Gaseous	131.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 111.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*° C.*	*Calories.*
July 27	80.95	0.978	807.2
Do	80.95	.982	810.6
July 28	80.95	.963	811.4

[a] Including 11 grams of the original wrapper.

Average large calories per kilogram of explosive, 809.7.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 31	63.50	49.25	14.25
Do	63.25	49.25	14.00
Do	63.50	49.75	13.75

Average compression, 14.0 millimeters (0.55 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*° C.*
Apr. 7	63	302	239	15
Do	63	300	237	15
Do	63	306	243	15

Average expansion of bore hole, 240 cubic centimeters (14.64 cubic inches).

MONOBEL NO. 5.

Explosive, Monobel No. 5.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, 1 inch.
Length of cartridge, 8 inches.
Average weight, 113 grams.
Cartridge had becn redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, corn.
Consistency, granular, fine, very dry, very soft, not cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, January 23, 1911.
Unit swing on this date, 3.05 inches.
Weight of charge, in grams, 265, 265, 265.
Swing, in inches, 3.00, 3.08, 3.01.
Average swing, in inches, 3.03.

3.03 : 3.05 : : 265 : (267).

Therefore the unit deflective charge of Monobel No. 5 is 267 grams.

GAS AND DUST GALLERY No. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Jan. 27	267	8.09	No ignition.	Feb. 4	267	.....	No ignition.
Do	267	8.14	Do.	Do	267	.....	Do.
Do	267	8.22	Do.	Do	267	.....	Do.
Do	267	8.09	Do.	Do	267	.....	Do.
Do	267	7.85	Do.	Do	267	.....	Do.
Do	267	7.91	Do.	Do	267	.....	Do.
Do	267	7.89	Do.	Do	267	.....	Do.
Do	267	8.05	Do.				
Do	267	8.20	Do.	TEST 4.			
Do	267	8.11	Do.				
				Feb. 1	[a]680	4.11	No ignition.
TEST 3.				Do	[a]680	3.96	Do.
				Do	[a]680	4.01	Do.
Feb. 4	267	.....	No ignition.	Do	[a]680	4.22	Do.
Do	267	.....	Do.	Do	[a]680	3.99	Do.
Do	267	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1913).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Jan. 9	21.85	45.0	2,059
Jan. 16	21.75	45.0	2,069
Jan. 17	21.90	45.0	2,055

Average rate of detonation, 2,061 meters (6,760 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1911).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 28	7.00	8.84	2.50	0.125
Do	10.00	12.63	3.50	.175
Do	8.00	10.10	2.75	.138

Average height of flame, 10.52 inches.
Average duration of flame, 0.146 millisecond.

Impact Test.

Date, July 25, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	10	1	No explosion.
19	1	Do.	11	1	Explosion.
16	1	Do.	10	4	No explosion.
12	1	Do.			

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 175 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 28	4	Did not explode.	Sept. 28	3	Did not explode.
Do	3	Do.	Do	3	Do.
Do	2	Exploded.			

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
July 29	215	1.03	27.00	84.38	A	
Do	215	1.03	28.25	88.28	A	86.72
Do	215	1.03	28.00	87.50	A	
July 28	215	1.03	26.00	81.25	B	
Do	215	1.03	26.00	81.25	B	80.47
Do	215	1.03	25.25	78.91	B	
July 27	215	1.03	23.00	71.88	C	
Do	215	1.03	22.75	71.09	C	72.40
July 28	215	1.03	23.75	74.22	C	

[a] Including 15 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=103.80$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.03$. $W=215$ grams.

$M=\frac{VPS}{W}=7{,}459$ kilograms per square centimeter (106,100 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 15 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, June 20, 1911.	Grams.
Solid	23.7
Liquid (water)	62.5
Gaseous	125.5

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.5 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 28	80.95	0.973	829.2
Do	80.95	1.005	856.7
Do	80.95	1.000	852.5

[a] Including 7.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 846.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 31	63.00	51.00	12.00
Do	63.50	51.25	12.25
Do	63.50	51.25	12.25

Average compression, 12.2 millimeters (0.48 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 6	63	210	147	15
Do	63	209	146	15
Do	63	210	147	15

Average expansion of bore hole, 147 cubic centimeters (8.97 cubic inches).

MONOBEL NO. 6.

Explosive, Monobel No. 6.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, $\frac{7}{8}$ inch.
Length of cartridge, 8 inches.
Average weight, 96 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.12.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 13, 1912.
Unit swing on this date, 3.37 inches.
Weight of charge, in grams, 220, 220, 220.
Swing, in inches, 3.31, 3.45, 3.45.
Average swing, in inches, 3.40.

3.40 : 3.37 : : 220 : (218).

Therefore the unit deflective charge of Monobel No. 6 is 218 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Feb. 13	218	8.12	No ignition.	Feb. 16	218	.....	No ignition.
Do	218	7.79	Do.	Do	218	.....	Do.
Do	218	8.27	Do.	Do	218	.....	Do.
Do	218	8.08	Do.	Do	218	.....	Do.
Do	218	8.33	Do.	Do	218	.....	Do.
Do	218	8.10	Do.	Do	218	.....	Do.
Do	218	8.21	Do.	Do	218	.....	Do.
Do	218	8.25	Do.				
Do	218	8.12	Do.	TEST 4.			
Do	218	8.40	Do.	Feb. 16	[a] 680	4.04	No ignition.
TEST 3.				Do	[a] 680	4.11	Do.
				Do	[a] 680	3.92	Do.
Feb. 15	218	.....	No ignition.	Feb. 17	[a] 680	4.07	Do.
Do	218	.....	Do.	Do	[a] 680	3.91	Do.
Do	218	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 14	14.20	45.0	3,169
Feb. 15	14.10	45.0	3,191
Feb. 16	14.35	45.0	3,136

Average rate of detonation, 3,165 meters (10,380 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
May 15	11.00	13.89	2.25	0.112
Do	9.25	11.68	3.50	.175
Do	16.50	20.84	6.00	.300

Average height of flame, 15.47 inches.
Average duration of flame, 0.196 millisecond.

Impact Test.

Date, April 19, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	23	1	Explosion.
25	1	Explosion.	22	3	No explosion.
23	1	No explosion.	22	1	Explosion.
24	4	Do.	21	5	No explosion.
24	1	Explosion.			

The maximum height at which no explosion occurs established at 21 centimeters (8.27 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 180 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 3	7	Did not explode.	May 3	4	Exploded.
Do	8	Exploded.	May 4	5	Did not explode.
Do	5	Did not explode.	Do	5	Do.

The minimum distance at which no explosion occurs established at 5 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 26	221	1.12	33.00	103.12	A	104.68
Do	221	1.12	34.50	107.81	A	
Do	221	1.12	33.00	103.12	A	
May 3	221	1.12	30.75	96.09	B	94.27
Do	221	1.12	29.50	92.19	B	
Do	221	1.12	30.25	94.53	B	
Do	221	1.12	29.25	91.41	C	92.71
May 4	221	1.12	29.50	92.19	C	
Do	221	1.12	30.25	94.53	C	

[a] Including 21 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=116.36$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.12$. $W=221$ grams.

$M=\frac{VPS}{W}=8{,}845$ kilograms per square centimeter (125,800 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 21 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, March 5, 1912.	Grams.
Solid	11.2
Liquid (water)	65.0
Gaseous	137.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 12	81.95	1.308	1,009.0
Mar. 13	81.95	1.326	1,114.2
Mar. 22	81.95	1.300	1,092.2

[a] Including 10.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,101.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 17	63.50	48.00	15.50
Do	63.25	48.00	15.25
Do	63.50	48.50	15.00

Average compression, 15.2 millimeters (0.60 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Apr. 25	62	342	280	19
Do	62	340	278	19
Apr. 26	62	340	278	18

Average expansion of bore hole, 279 cubic centimeters (17.02 cubic inches).

MONOBEL NO. 7.

Explosive, Monobel No. 7.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the E. I. du Pont de Nemours Powder Co.
Physical examination:
Diameter of cartridge, $\frac{7}{8}$ inch.
Length of cartridge, 8 inches.
Average weight, 93 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.09.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, February 13, 1912.

Unit swing on this date, 3.37 inches.

Weight of charge, in grams, 220, 220, 220.

Swing, in inches, 3.57, 3.46, 3.53.

Average swing, in inches, 3.52.

3.52 : 3.37 : : 220 : (211).

Therefore the unit deflective charge of Monobel No. 7 is 211 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Feb. 14	211	8.27	No ignition.	Feb. 15	211	.....	No ignition.
Do	211	8.32	Do.	Do	211	.....	Do.
Do	211	8.30	Do.	Do	211	.....	Do.
Do	211	8.47	Do.	Do	211	.....	Do.
Do	211	8.25	Do.	Do	211	.....	Do.
Do	211	8.27	Do.	Do	211	.....	Do.
Do	211	8.54	Do.	Do	211	.....	Do.
Do	211	8.28	Do.				
Feb. 15	211	8.08	Do.	TEST 4.			
Do	211	8.25	Do.				
				Feb. 17	[a] 680	4.08	No ignition.
TEST 3.				Do	[a] 680	4.15	Do.
				Do	[a] 680	4.03	Do.
Feb. 15	211	.....	No ignition.	Do	[a] 680	4.08	Do.
Do	211	.....	Do.	Do	[a] 680	4.25	Do.
Do	211	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Feb. 14	14.85	45.0	3,030
Feb. 16	14.60	45.0	3,082
Do	14.60	45.0	3,082

Average rate of detonation, 3,065 meters (10,050 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
May 11	11.75	14.84	6.00	0.300
Do	11.75	14.84	5.50	.275
Do	13.00	16.42	5.50	.275

Average height of flame, 15.37 inches.

Average duration of flame, 0.283 millisecond.

Impact Test.

Date, April 20, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	No explosion.	32	1	Explosion.
25	1	Do.	31	1	No explosion.
35	1	Explosion.	31	1	Explosion.
32	1	No explosion.	30	5	No explosion.
33	1	Explosion.			

The maximum height at which no explosion occurs established at 30 centimeters (11.81 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 175 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
May 4.....	4	Exploded.	May 4.....	5	Exploded.
Do.....	6	Did not explode.	May 6.....	6	Did not explode.
Do.....	5	Do.	Do.....	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Apr. 26.....	220	1.09	33.00	103.12	A	
Do.....	220	1.09	32.75	102.34	A	102.86
Do.....	220	1.09	33.00	103.12	A	
May 6.....	220	1.09	30.50	95.31	B	
Do.....	220	1.09	30.25	94.53	B	94.01
Do.....	220	1.09	29.50	92.19	B	
May 7.....	220	1.09	30.50	95.31	B	
Do.....	220	1.09	31.00	96.88	B	95.57
Do.....	220	1.09	30.25	94.53	B	

[a] Including 20 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=108.72$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.09$. $W=220$ grams.

$M=\frac{VPS}{W}=8{,}080$ kilograms per square centimeter (114,920 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 20 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, March 6, 1912.

	Grams.
Solid	10.0
Liquid (water)	69.5
Gaseous	134.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*° C.*	*Calories.*
Mar. 20	81.95	1.238	1,044.5
Feb. 24	81.95	1.234	1,041.1
Do	81.95	1.263	1,065.7

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 1,050.4.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Apr. 17	63.25	47.75	15.50
Do	63.25	47.75	15.50
Do	63.25	48.00	15.25

Average compression, 15.4 millimeters (0.61 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*° C.*
Apr. 25	62	340	278	19
Do	62	334	272	19
Apr. 29	62	344	282	16

Average expansion of bore hole, 277 cubic centimeters (16.90 cubic inches).

NITRO LOW-FLAME NO. 1.

Explosive, Nitro low-flame No. 1.
Class 4, nitroglycerin.
Manufactured by the Nitro Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 218 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.22.
Color of explosive, straw.
Consistency, powdered, very fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 15, 1910.

Unit swing on this date, 3.32 inches.

Weight of charge, in grams, 300, 300, 300.

Swing, in inches, 3.34, 3.34, 3.20.

Average swing, in inches, 3.29.

3.29 : 3.32 : : 300 : (303).

Therefore the unit deflective charge of Nitro low-flame No. 1 is 303 grams.

GAS AND DUST GALLERY No. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 19	303	8.22	No ignition.	Sept. 20	303	.....	No ignition.
Do	303	8.16	Do.	Do	303	.....	Do.
Do	303	8.20	Do.	Do	303	.....	Do.
Do	303	8.16	Do.	Do	303	.....	Do.
Do	303	8.30	Do.	Do	303	.....	Do.
Do	303	8.30	Do.	Do	303	.....	Do.
Do	303	8.07	Do.	Do	303	.....	Do.
Do	303	8.24	Do.				
Do	303	8.26	Do.	TEST 4.			
Do	303	7.92	Do.				
				Sept. 16	[a] 680	4.18	No ignition.
TEST 3.				Do	[a] 680	4.22	Do.
				Do	[a] 680	4.22	Do.
Sept. 20	303	.....	No ignition.	Do	[a] 680	4.13	Do.
Do	303	.....	Do.	Do	[a] 680	4.18	Do.
Do	303	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 16	10.70	43.5	4,065
Do	10.80	43.5	4,028
Do	10.50	43.5	4,143

Average rate of detonation, 4,079 meters (13,380 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 23	12.50	15.79	5.50	0.275
Do	14.75	18.63	5.50	.275
Sept. 30	11.25	14.21	4.75	.238

Average height of flame, 16.21 inches.

Average duration of flame, 0.263 millisecond.

IMPACT TEST.

Date, February 18, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
26	1	Explosion.	20	1	Explosion.
20	1	No explosion.	16	1	Do.
24	1	Explosion.	14	2	No explosion.
22	1	No explosion.	14	1	Explosion.
23	1	Do.	13	2	No explosion.
23	1	Explosion.	13	1	Explosion.
22	1	Do.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 196 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 17	8	Did not explode.	Feb. 18	5	Exploded.
Do.	4	Exploded.	Do.	6	Did not explode.
Do.	6	Did not explode.	Do.	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 3	214.8	1.22	25.00	62.50	A	62.08
Mar. 7	214.8	1.22	24.25	60.62	A	
Mar. 13	214.8	1.22	25.25	63.12	A	
Mar. 2	214.8	1.22	25.00	62.50	B	62.29
Do	214.8	1.22	24.75	61.88	B	
Do	214.8	1.22	25.00	62.50	B	
Mar. 3	214.8	1.22	21.50	53.75	C	54.58
Do	214.8	1.22	22.00	55.00	C	
Do	214.8	1.22	22.00	55.00	C	

[a] Including 14.8 grams of the original wrapper.

P=1.911A+0.5B−1.411C=72.77 kilograms per square centimeter.

V=15,000 cubic centimeters. S=1.22. W=214.8 grams.

$M=\frac{VPS}{W}$=6,200 kilograms per square centimeter (88,190 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14.8 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, February 20, 1911.

	Grams.
Solid	60.5
Liquid (water)	11.5
Gaseous	134.3

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 107.4 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 28	80.95	0.698	593.2
Do	80.95	.714	606.9
Do	80.95	.704	598.3

[a] Including 7.4 grams of the original wrapper.

Average large calories per kilogram of explosive, 599.5.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 10	63.50	45.25	18.25
Do	63.50	45.50	18.00
Do	63.50	45.25	18.25

Average compression, 18.2 millimeters (0.72 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams,

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 27	63	215	152	15
Do	63	212	149	15
Do	63	219	156	15

Average expansion of bore hole, 152 cubic centimeters (9.27 cubic inches).

NITRO LOW-FLAME NO. 2.

Explosive, Nitro low-flame No. 2.
Class 4, nitroglycerin.
Manufactured by the Nitro Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 244 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.31.
Color of explosive, straw.
Consistency, powdered, very fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, September 15, 1910.

Unit swing on this date, 3.32 inches.

Weight of charge, in grams, 285, 285, 285.

Swing, in inches, 3.10, 3.16, 3.25.

Average swing, in inches, 3.17.

3.17 : 3.32 : : 285 : (298).

Therefore the unit deflective charge of Nitro low-flame No. 2 is 298 grams.

GAS AND DUST GALLERY NO. 1.

Date (1910).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 19	298	8.07	No ignition.	Sept. 21	298	.....	No ignition.
Do	298	8.31	Do.	Do	298	.....	Do.
Do	298	7.94	Do.	Do	298	.....	Do.
Do	298	8.25	Do.	Do	298	.....	Do.
Do	298	7.97	Do.	Do	298	.....	Do.
Do	298	8.07	Do.	Do	298	.....	Do.
Do	298	8.07	Do.	Do	298	.....	Do.
Do	298	8.05	Do.				
Do	298	8.07	Do.	TEST 4.			
Sept. 20	298	8.07	Do.				
				Sept. 17	[a] 680	4.18	No ignition.
TEST 3.				Do	[a] 680	4.47	Do.
				Do	[a] 680	4.32	Do.
Sept. 20	298	.....	No ignition.	Do	[a] 680	4.27	Do.
Do	298	.....	Do.	Oct. 12	[a] 680	3.94	Do.
Sept. 21	298	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Sept. 16	9.57	43.5	4,545
Sept. 19	10.00	43.5	4,350
Do	9.84	43.5	4,421

Average rate of detonation, 4,439 meters (14,560 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 30	17.75	22.42	7.25	0.362
Do	20.25	25.58	10.50	.525
Do	22.75	28.74	11.00	.550

Average height of flame, 25.58 inches.

Average duration of flame, 0.479 millisecond.

Impact Test.

Date, February 18, 1911.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
16	1	No explosion.	20	1	Explosion.
18	1	Do.	18	1	Do.
20	1	Do.	16	1	Do.
24	1	Explosion.	15	1	Do.
22	1	Do.	14	5	No explosion.

The maximum height at which no explosion occurs established at 14 centimeters (5.51 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 211 grams.

Date (1911).	Distance separating cartridges.	Result, upper cartridge.	Date (1911).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Feb. 18.....	6	No explosion.	Feb. 18.....	6	No explosion.
Do.....	5	Explosion.	Feb. 20.....	6	Do.

The minimum distance at which no explosion occurs established at 6 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1911).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 14.....	212	1.31	27.00	67.50	A	
Do.....	212	1.31	27.00	67.50	A	67.71
Do.....	212	1.31	27.25	68.12	A	
Do.....	212	1.31	25.50	63.75	B	
Do.....	212	1.31	25.00	62.50	B	63.54
Do.....	212	1.31	25.75	64.38	B	
Do.....	212	1.31	23.50	58.75	C	
Do.....	212	1.31	24.25	60.62	C	59.37
Do.....	212	1.31	23.50	58.75	C	

[a] Including 12 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=77.39$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.31$. $W=212$ grams.

$M=\frac{VPS}{W}=7{,}173$ kilograms per square centimeter (102,030 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 12 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, February 21, 1911.

	Grams.
Solid	64.3
Liquid (water)	14.3
Gaseous	124.7

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.0 grams.[a]

Date (1911).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Mar. 1	80.95	0.746	642.9
Do	80.95	.777	669.9
Do	80.95	.752	648.1

[a] Including 6 grams of the original wrapper.

Average large calories per kilogram of explosive, 653.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1911).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 10	63.50	45.25	18.25
Do	63.50	44.75	18.75
Do	63.50	44.50	19.00

Average compression, 18.7 millimeters (0.74 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1911).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 27	63	232	169	15
Do	63	226	163	15
Do	63	230	167	15

Average expansion of bore hole, 166 cubic centimeters (10.13 cubic inches).

TROJAN COAL POWDER H.

Explosive, Trojan coal powder H.
Class 3, organic nitrate.
Manufactured by the Pennsylvania Trojan Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8½ inches.
Average weight, 506 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.40.
Color of explosive, white.
Consistency, powdered, very fine, very dry, very soft, not cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, October 18, 1911.

Unit swing on this date, 3.30 inches.

Weight of charge, in grams, 400, 400, 400.

Swing, in inches, 3.40, 3.33, 3.31.

Average swing, in inches, 3.35.

3.35 : 3.30 : : 400 : (394).

Therefore the unit deflective charge of Trojan coal powder H is 394 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Oct. 19	394	8.35	No ignition.	Nov. 3	394	.....	No ignition.
Do	394	8.39	Do.	Do	394	.....	Do.
Do	394	8.40	Do.	Do	394	.....	Do.
Do	394	8.12	Do.	Do	394	.....	Do.
Do	394	7.83	Do.	Do	394	.....	Do.
Do	394	7.91	Do.	Do	394	.....	Do.
Do	394	7.90	Do.	Do	394	.....	Do.
Oct. 20	394	8.02	Do.				
Do	394	8.35	Do.	TEST 4.			
Oct. 21	394	8.06	Do.	Oct. 24	[a] 680	4.17	No ignition.
TEST 3.				Do	[a] 680	4.34	Do.
				Do	[a] 680	4.17	Do.
Nov. 3	394	.....	No ignition.	Do	[a] 680	4.16	Do.
Do	394	.....	Do.	Do	[a] 680	3.90	Do.
Do	394	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 18	15.60	44.0	2,821
Do	15.25	44.0	2,885
Do	15.12	44.0	2,910

Average rate of detonation, 2,872 meters (9,420 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 7	19.50	24.63	10.25	0.513
Do	18.00	22.74	9.00	.450
Do	15.75	19.89	8.00	.400

Average height of flame, 22.42 inches.

Average duration of flame, 0.454 millisecond.

IMPACT TEST.

Date, February 21, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	2	No explosion.
10	1	No explosion.	15	1	Explosion.
15	1	Do.	14	2	No explosion.
18	1	Explosion.	14	1	Explosion.
17	1	Do.	13	1	Do.
16	1	Do.	12	5	No explosion.

The maximum height at which no explosion occurs established at 12 centimeters (4.72 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 225 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Mar. 18	4	Did not explode.	Mar. 18	3	Did not explode.
Do	2	Exploded.	Do	3	Do.
Do	3	Did not explode.			

The minimum distance at which no explosion occurs established at 3 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.60 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
		Grams.	*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 29	207.2	1.40	23.75	39.58	A	
Do	207.2	1.40	24.00	40.00	A	39.30
Mar. 1	207.2	1.40	23.00	38.33	A	
Feb. 29	207.2	1.40	22.75	37.92	B	
Do	207.2	1.40	23.25	38.75	B	37.92
Do	207.2	1.40	22.25	37.08	B	
Do	207.2	1.40	21.75	36.25	C	
Do	207.2	1.40	22.00	36.67	C	36.39
Do	207.2	1.40	21.75	36.25	C	

[a] Including 7.2 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=42.72$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.40$. $W=207.2$ grams.

$M=\frac{VPS}{W}=4{,}330$ kilograms per square centimeter (61,590 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 7 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, November 24, 1911.

	Grams.
Solid	93.9
Liquid (water)	9.4
Gaseous	92.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 103.6 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 13	81.95	0.768	684.8
Do	81.95	.734	654.1
Do	81.95	.749	667.7

[a] Including 3.6 grams of the original wrapper.

Average large calories per kilogram of explosive, 668.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 27	63.50	47.50	16.00
Do	63.50	48.00	15.50
Do	63.50	48.25	15.25

Average compression, 15.6 millimeters (0.61 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912.)	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 21	62	170	108	15
Do	62	168	106	15
Do	62	170	108	15

Average expansion of bore hole, 107 cubic centimeters (6.53 cubic inches).

TROJAN COAL POWDER I.

Explosive, Trojan coal powder I.
Class 3, organic nitrate.
Manufactured by the Pennsylvania Trojan Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8¼ inches.
Average weight, 501 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.38.
Color of explosive, white.
Consistency, powdered; very fine, very dry, very soft, not cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, October 18, 1911.

Unit swing used on this date, 3.30 inches.

Weight of charge, in grams, 400, 400, 400.

Swing, in inches, 3.22, 3.24, 3.20.

Average swing, in inches, 3.22.

3.22 : 3.30 : : 400 : (410).

Therefore the unit deflective charge of the Trojan coal powder I is 410 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Oct. 21	410	8.31	No ignition.	Nov. 3	410	.....	No ignition.
Do	410	8.33	Do.	Do	410	.....	Do.
Do	410	7.99	Do.	Do	410	.....	Do.
Do	410	7.96	Do.	Do	410	.....	Do.
Do	410	7.88	Do.	Do	410	.....	Do.
Do	410	8.01	Do.	Do	410	.....	Do.
Do	410	8.16	Do.	Do	410	.....	Do.
Oct. 23	410	8.15	Do.	TEST 4.			
Do	410	7.95	Do.				
Do	410	7.98	Do.	Oct. 24	[a] 680	4.01	No ignition.
TEST 3.				Do	[a] 680	4.35	Do.
				Do	[a] 680	4.14	Do.
Nov. 2	410	.....	No ignition.	Oct. 25	[a] 680	4.22	Do.
Do	410	.....	Do.	Do	[a] 680	4.08	Do.
Do	410	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 19	15.20	44.0	2,895
Do	14.80	44.0	2,973
Oct. 20	15.00	44.0	2,933

Average rate of detonation, 2,934 meters (9,620 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 7	17.00	21.47	8.00	0.400
Do	18.00	22.74	8.00	.400
Do	16.50	20.84	9.00	.450

Average height of flame, 21.68 inches.

Average duration of flame, 0.417 millisecond.

Impact Test.

Date, February 21, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
12	1	No explosion.	21	2	No explosion.
15	1	Do.	21	1	Explosion.
18	1	Do.	20	5	No explosion.
22	1	Explosion.			

The maximum height at which no explosion occurs established at 20 centimeters (7.87 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 222 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Mar. 6	3	Did not explode.	Mar. 6	4	Did not explode.
Do	2	Exploded.	Mar. 7	4	Do.
Do	3	Do.	Do	4	Do.

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.60 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 1	208	1.38	26.50	44.17	A	44.17
Do	208	1.38	26.50	44.17	A	
Do	208	1.38	26.50	44.17	A	
Do	208	1.38	23.75	39.58	B	39.30
Do	208	1.38	23.75	39.58	B	
Mar. 2	208	1.38	23.25	38.75	B	
Do	208	1.38	23.00	38.33	C	37.50
Mar. 4	208	1.38	22.50	37.50	C	
Do	208	1.38	22.00	36.67	C	

[a] Including 8 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=51.15$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.38$. $W=208$ grams.

$M=\frac{VPS}{W}=5{,}090$ kilograms per square centimeter (72,400 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 8 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, December 5, 1911.	Grams.
Solid	84.8
Liquid (water)	10.8
Gaseous	92.1

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 104.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 17	81.95	0.873	776.6
Do	81.95	.882	784.7
Feb. 19	81.95	.916	815.3

[a] Including 4 grams of the original wrapper.

Average large calories per kilogram of explosive, 792.2.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 27	63.50	48.75	14.75
Do	63.50	49.25	14.25
Do	63.50	49.25	14.25

Average compression, 14.4 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 21	62	168	106	15
Do	62	164	102	15
Do	62	166	104	15

Average expansion of bore hole, 104 cubic centimeters (6.34 cubic inches).

TROJAN COAL POWDER J.

Explosive, Trojan coal powder J.
Class 3, organic nitrate.
Manufactured by the Pennsylvania Trojan Powder Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8¼ inches.
Average weight, 471 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.29.
Color of explosive, white.
Consistency, powdered, very fine, very dry, very soft, not cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, October 18, 1911.
Unit swing on this date, 3.30 inches.
Weight of charge, in grams, 350, 350, 350.
Swing, in inches, 3.14, 3.21, 3.28.
Average swing, in inches, 3.21.

3.21 : 3.30 : : 350 : (360).

Therefore the unit deflective charge of Trojan coal powder J is 360 grams.

GAS AND DUST GALLERY NO. 1.

Date (1911).	Weight of charge.	Methane and ethane.	Result.	Date (1911).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Oct. 23	360	7.93	No ignition.	Nov. 2	360	.....	No ignition.
Do	360	7.99	Do.	Do	360	.....	Do.
Do	360	8.06	Do.	Do	360	.....	Do.
Do	360	8.05	Do.	Do	360	.....	Do.
Do	360	8.14	Do.	Do	360	.....	Do.
Do	360	8.17	Do.	Do	360	.....	Do.
Do	360	8.12	Do.	Do	360	.....	Do.
Do	360	7.80	Do.				
Do	360	7.86	Do.	TEST 4.			
Do	360	7.82	Do.				
				Oct. 25	[a] 680	3.98	No ignition.
TEST 3.				Do	[a] 680	3.94	Do.
				Do	[a] 680	4.18	Do.
Nov. 2	360	.....	No ignition.	Do	[a] 680	3.95	Do.
Do	360	.....	Do.	Do	[a] 680	4.06	Do.
Do	360	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1911).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Oct. 20	13.74	44.0	3,202
Do	13.40	44.0	3,284
Do	13.73	44.0	3,205

Average rate of detonation, 3,230 meters (10,590 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Mar. 7	20.50	25.89	10.50	0.525
Do	17.00	21.47	8.50	.425
Do	22.00	27.79	11.50	.575

Average height of flame, 25.05 inches.
Average duration of flame, 0.508 millisecond.

IMPACT TEST.

Date, February 23, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
20	1	Explosion.	15	1	No explosion.
15	1	No explosion.	15	1	Explosion.
18	1	Do.	14	1	Do.
19	1	Explosion.	13	2	No explosion.
18	1	Do.	13	1	Explosion.
17	1	No explosion.	12	1	Do.
17	1	Explosion.	11	5	No explosion.
16	1	Do.			

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 207 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Mar. 9	3	Exploded.	Mar. 9	5	Did not explode.
Do	4	Did not explode.	Mar. 12	5	Do.
Do	4	Do.	Do	5	Do.
Do	4	Exploded.			

The minimum distance at which no explosion occurs established at 5 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.60 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Mar. 6	208	1.29	30.50	50.83	A	50.00
Do	208	1.29	30.50	50.83	A	
Do	208	1.29	29.00	48.33	A	
Mar. 5	208	1.29	26.75	44.58	B	45.00
Do	208	1.29	26.50	44.17	B	
Mar. 6	208	1.29	27.75	46.25	B	
Mar. 7	208	1.29	27.00	45.00	C	45.00
Do	208	1.29	27.25	45.42	C	
Do	208	1.29	26.75	44.58	C	

[a] Including 8 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=54.56$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.29$. $W=208$ grams.

$M=\frac{VPS}{W}=5{,}076$ kilograms per square centimeter (72,200 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 8 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 6, 1911.

	Grams.
Solid	81.7
Liquid (water)	14.5
Gaseous	97.1

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 104.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Feb. 19	81.95	0.792	703.7
Do	81.95	.804	714.5
Do	81.95	.827	735.2

[a] Including 4 grams of the original wrapper.

Average large calories per kilogram of explosive, 717.8.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 27	63.50	47.50	16.00
Do	63.50	48.00	15.50
Do	63.50	48.00	15.50

Average compression, 15.7 millimeters (0.62 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Feb. 23	62	190	128	15
Do	62	186	124	15
Do	62	192	130	15

Average expansion of bore hole, 127 cubic centimeters (7.75 cubic inches).

TUNNELITE B.

Explosive, Tunnelite B.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 164 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.94.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, April 15, 1912.
Unit swing on this date, 3.37 inches.
Weight of charge, in grams, 240, 240, 240.
Swing, in inches, 3.23, 3.28, 3.23.
Average swing, in inches, 3.25.

3.25 : 3.37 : : 240 : (249).

Therefore the unit deflective charge of Tunnelite B is 249 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Apr. 16	249	8.46	No ignition.	Apr. 24	249	.....	No ignition.
Do	249	8.37	Do.	Do	249	.....	Do.
Do	249	8.39	Do.	Do	249	.....	Do.
Apr. 17	249	8.07	Do.	Do	249	.....	Do.
Do	249	8.22	Do.	Do	249	.....	Do.
Do	249	8.43	Do.	Do	249	.....	Do.
Do	249	8.25	Do.	Do	249	.....	Do.
Do	249	8.10	Do.				
Do	249	8.34	Do.	TEST 4.			
Do	249	8.14	Do.				
				Apr. 23	[a] 680	4.04	No ignition.
TEST 3.				Do	[a] 680	4.18	Do.
				Do	[a] 680	4.18	Do.
Apr. 24	249	.....	No ignition.	Do	[a] 680	4.17	Do.
Do	249	.....	Do.	Do	[a] 680	4.04	Do.
Do	249	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
May 2	15.05	45.0	2,990
Do	14.45	44.5	3,080
Do	14.65	45.0	3,072

Average rate of detonation, 3,047 meters (9,990 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 7	7.50	9.47	2.00	0.100
Sept. 10	6.25	7.89	1.50	.075
Do	5.75	7.26	1.00	.050

Average height of flame, 8.21 inches.
Average duration of flame, 0.075 millisecond.

Impact Test.

Date, September 6, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
25	1	No explosion.	15	1	Explosion.
30	1	Explosion.	10	1	No explosion.
27	1	Do.	14	1	Explosion.
25	1	Do.	13	5	No explosion.
20	1	Do.			

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 151 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 6	3	Did not explode.	Sept. 6	2	Exploded.
Do	2	Do.	Do	3	Did not explode.
Do	1	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 8	219	0.94	30.00	93.75	A	
Do	219	.94	30.00	93.75	A	94.53
Do	219	.94	30.75	96.09	A	
Aug. 7	219	.94	28.00	87.50	B	
Do	219	.94	28.25	88.28	B	88.54
Do	219	.94	28.75	89.84	B	
Aug. 6	219	.94	28.00	87.50	C	
Aug. 7	219	.94	28.25	88.28	C	87.50
Do	219	.94	27.75	86.72	C	

[a] Including 19 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=101.45$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=0.94$. $W=219$ grams.

$M=\frac{VPS}{W}=6{,}532$ kilograms per square centimeter (92,910 pounds per square inch).

Products of Combustion from 200 Grams of the Explosive and 19 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, April 29, 1912.	Grams.
Solid	24.6
Liquid (water)	63.0
Gaseous	126.1

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 109.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 24	81.95	1.100	931.4
Do	81.95	1.108	938.3
July 25	81.95	1.120	948.5

[a] Including 9.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 939.4.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Sept. 24	63.25	49.75	13.50
Do	63.25	49.75	13.50
Do	63.25	50.00	13.25

Average compression, 13.4 millimeters (0.53 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Sept. 21	62	306	244	16
Do	62	305	243	16
Do	62	312	250	16

Average expansion of bore hole, 246 cubic centimeters (15.01 cubic inches).

TUNNELITE C.

Explosive, Tunnelite C.
Class 1*a*, ammonium nitrate, containing nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 178 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.02.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Unit deflective charge as determined by the ballistic pendulum:

Date, April 15, 1912.

Unit swing on this date, 3.37 inches.

Weight of charge, in grams, 250, 250, 250.

Swing, in inches, 3.25, 3.40, 3.24.

Average swing, in inches, 3.30.

3.30 : 3.37 : : 250 : (255).

Therefore the unit deflective charge of Tunnelite C is 255 grams.

GAS AND DUST GALLERY NO. 1.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—CON.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Apr. 17	255	8.07	No ignition.	Apr. 24	255	.....	No ignition.
Do	255	7.87	Do.	Do	255	.....	Do.
Do	255	7.77	Do.	Do	255	.....	Do.
Do	255	7.87	Do.	Do	255	.....	Do.
Do	255	8.51	Do.	Do	255	.....	Do.
Do	255	7.97	Do.	Do	255	.....	Do.
Do	255	7.97	Do.	Do	255	.....	Do.
Do	255	8.33	Do.				
Apr. 18	255	8.36	Do.	TEST 4.			
Do	255	8.09	Do.				
				Apr. 23	[a] 680	4.06	No ignition.
TEST 3.				Do	[a] 680	4.17	Do.
				Do	[a] 680	4.20	Do.
Apr. 24	255	.....	No ignition.	Do	[a] 680	4.19	Do.
Do	255	.....	Do.	Do	[a] 680	4.22	Do.
Do	255	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.

Electric detonator, No. 7.

Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
May 2	14.80	45.0	3,041
May 3	14.35	44.5	3,101
Do	14.20	43.5	3,063

Average rate of detonation, 3,068 meters (10,060 feet) per second.

IMPACT TEST.

Date, August 19, 1912.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
40	1	No explosion.	65	1	Explosion.
50	1	Do.	62	1	No explosion.
60	1	Do.	64	5	Do.
70	1	Explosion.			

The maximum height at which no explosion occurs established at 64 centimeters (25.2 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 164 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Sept. 7	3	Did not explode.	Sept. 7	2	Exploded.
Do	2	Do.	Do	3	Did not explode.
Do	1	Exploded.	Do	3	Do.

The minimum distance at which no explosion occurs established at 3 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1912).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Aug. 19	218	1.02	30.25	94.53	A	96.61
Do	218	1.02	30.75	96.09	A	
Do	218	1.02	31.75	99.22	A	
Aug. 23	218	1.02	28.00	87.50	B	87.50
Do	218	1.02	28.00	87.50	B	
Do	218	1.02	28.00	87.50	B	
Aug. 21	218	1.02	27.50	85.94	C	85.94
Do	218	1.02	27.75	86.72	C	
Do	218	1.02	27.25	85.16	C	

[a] Including 18 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=107.11$ kilograms per square centimeter.

$V=15{,}000$ cubic centimeters. $S=1.02$. $W=218$ grams.

$M=\frac{VPS}{W}=7{,}517$ kilograms per square centimeter (106,920 pounds per square inch).

Products of Combustion From 200 Grams of the Explosive and 19 Grams of the Original Wrapper.

[Analyst, A. L. Hyde.]

Date, April 30, 1912.	Grams.
Solid	24.7
Liquid (water)	61.5
Gaseous	125.2

Large Calories Developed by 1 Kilogram of the Explosive.

Charge, 109.0 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
July 25	81.95	1.139	969.2
Do	81.95	1.100	935.7
Do	81.95	1.100	935.7

[a] Including 9 grams of the original wrapper.

Average large calories per kilogram of explosive, 946.9.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1912).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Aug. 14	63.25	48.75	14.50
Aug. 15	63.25	48.50	14.75
Do	63.50	49.25	14.25

Average compression, 14.5 millimeters (0.57 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1912).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	°C.
Sept. 27	62	270	208	15
Do	62	272	210	15
Do	62	277	215	15

Average expansion of bore hole, 211 cubic centimeters (12.87 cubic inches).

TUNNELITE NO. 5.

Explosive, Tunnelite No. 5.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1½ inches.
Length of cartridge, 8 inches.
Average weight, 220 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.06.
Color of explosive, ecru with green spots.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, August 27, 1909.
Unit swing on this date, 2.61 inches.
Weight of charge, in grams, 315, 315, 315.
Swing, in inches, 2.55, 2.54, 2.45.
Average swing, in inches, 2.51.

$$2.51 : 2.61 : : 315 : (328).$$

Therefore the unit deflective charge of Tunnelite No. 5 is 328 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 31	328	8.21	No ignition.	Sept. 1	328	.....	No ignition.
Do	328	8.41	Do.	Do	328	.....	Do.
Do	328	8.51	Do.	Do	328	.....	Do.
Do	328	8.60	Do.	Do	328	.....	Do.
Do	328	8.16	Do.	Do	328	.....	Do.
Do	328	8.45	Do.	Do	328	.....	Do.
Do	328	8.50	Do.	Do	328	.....	Do.
Do	328	8.45	Do.				
Do	328	8.45	Do.	TEST 4.			
Sept. 2	328	8.33	Do.				
				Aug. 13	[a] 680	4.35	No ignition.
TEST 3.				Do	[a] 680	4.16	Do.
				Do	[a] 680	4.16	Do.
Aug. 31	328	.....	No ignition.	Do	[a] 680	3.92	Do.
Sept. 1	328	.....	Do.	Do	[a] 680	4.00	Do.
Do	328	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909–10).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Aug. 27	16.90	43.0	2,544
Aug. 28	16.60	43.0	2,590
June 2	16.85	43.0	2,552

Average rate of detonation, 2,562 meters (8,400 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 12	19.50	24.63	8.50	0.425
Do	11.25	14.21	6.50	.325
Do	16.50	20.84	7.75	.388

Average height of flame, 19.89 inches.
Average duration of flame, 0.379 millisecond.

Impact Test.

Date, May 5, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
14	1	No explosion	13	1	Explosion.
18	1	Explosion.	12	1	No explosion.
16	1	Do.	12	1	Explosion.
15	1	Do.	11	1	Do.
14	3	No explosion.	10	5	No explosion.
14	1	Explosion.			

The maximum height at which no explosion occurs established at 10 centimeters (3.94 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 180 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 3.....	10	Exploded.	Dec. 4.....	16	Did not explode.
Dec. 4.....	12	Do.	Do.....	16	Exploded.
Do.....	14	Did not explode.	Do.....	17	Did not explode.
Do.....	13	Exploded.	Do.....	17	Exploded.
Do.....	14	Do.	Do.....	18	Did not explode.
Do.....	15	Did not explode.	Do.....	18	Exploded.
Do.....	15	Do.	Do.....	19	Did not explode.
Do.....	15	Exploded.	Do.....	19	Do.
Do.....	16	Did not explode.	Do.....	19	Do.

The minimum distance at which no explosion occurs established at 19 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 31.....	213	1.12	19.75	61.72	A	62.24
Do.....	213	1.12	20.00	62.50	A	
Do.....	213	1.12	20.00	62.50	A	
Do.....	213	1.12	20.00	62.50	B	61.46
Do.....	213	1.12	19.75	61.72	B	
Feb. 13.....	213	1.12	19.25	60.16	B	
Jan. 31.....	213	1.12	18.50	57.81	C	57.03
Do.....	213	1.12	18.00	56.25	C	
Do.....	213	1.12	18.25	57.03	C	

[a] Including 13 grams of the original wrapper.

$P = 1.911A + 0.5B - 1.411C = 69.20$ kilograms per square centimeter.

$V = 15{,}000$ cubic centimeters. $S = 1.12$. $W = 213$ grams.

$M = \frac{VPS}{W} = 5{,}458$ kilograms per square centimeter (77,630 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 13 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 15, 1910.	Grams.
Solid	82.9
Liquid (water)	12.1
Gaseous	113.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 12	81.95	0.786	662.0
Do	81.95	.771	668.8
Do	81.95	.791	686.4

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 679.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
June 2	63.50	54.25	9.25
Do	63.50	54.00	9.50
Do	63.50	54.25	9.25

Average compression, 9.3 millimeters (0.37 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	183	120	15
Do	63	186	123	15
Do	63	188	125	15

Average expansion of bore hole, 123 cubic centimeters (7.50 cubic inches).

TUNNELITE NO. 6.

Explosive, Tunnelite No. 6.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 202 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.97.
Color of explosive, Nile green.
Consistency, granular, fine, dry, soft, moderately cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, August 27, 1909.
Unit swing on this date, 2.61 inches.
Weight of charge, in grams, 305, 305, 305.
Swing, in inches, 2.57, 2.58, 2.67.
Average swing, in inches, 2.61.

2.61 : 2.61 : : 305 : (305).

Therefore the unit deflective charge of Tunnelite No. 6 is 305 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Sept. 2	305	7.93	No ignition.	Sept. 3	305	.....	No ignition.
Do	305	8.30	Do.	Do	305	.....	Do.
Do	305	8.04	Do.	Do	305	.....	Do.
Do	305	8.20	Do.	Do	305	.....	Do.
Do	305	8.21	Do.	Do	305	.....	Do.
Do	305	8.30	Do.	Do	305	.....	Do.
Do	305	8.23	Do.	Do	305	.....	Do.
Do	305	8.33	Do.				
Do	305	8.06	Do.	TEST 4.			
Do	305	8.21	Do.	Sept. 1	[a] 680	4.20	No ignition.
				Do	[a] 680	4.53	Do.
TEST 3.				Do	[a] 680	4.32	Do.
Sept. 2	305	.....	No ignition.	Do	[a] 680	4.26	Do.
Do	305	.....	Do.	Do	[a] 680	4.37	Do.
Sept. 3	305	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 6	14.46	43.0	2,974
June 7	14.78	43.0	2,909
Do	14.54	43.0	2,957

Average rate of detonation, 2,947 meters (9,670 feet) per second.

FLAME TEST.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 12	25.00	31.58	12.00	0.600
Do	18.00	22.74	8.50	.425
Do	20.00	25.26	8.50	.425

Average height of flame, 26.53 inches.
Average duration of flame, 0.483 millisecond.

IMPACT TEST.

Date, May 6, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
12	1	No explosion.	14	1	Explosion.
14	1	Do.	13	1	Do.
16	1	Explosion.	12	1	Do.
15	1	Do.	11	5	No explosion.

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

EXPLOSION-BY-INFLUENCE TEST.

Weight of each cartridge, 167 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 3	5	Exploded.	Dec. 3	13	Exploded.
Do	9	Do.	Do	14	Do.
Do	12	Do.	Do	15	Did not explode.
Do	14	Did not explode.	Do	15	Do.
Do	13	Do.	Do	15	Do.

The minimum distance at which no explosion occurs established at 15 inches.

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Jan. 31	213	1.04	21.75	67.97	A	67.19
Do	213	1.04	21.75	67.97	A	
Feb. 1	213	1.04	21.00	65.62	A	
Do	213	1.04	20.50	64.06	B	64.06
Do	213	1.04	20.75	64.84	B	
Do	213	1.04	20.25	63.28	B	
Feb. 3	213	1.04	19.75	61.72	C	62.24
Do	213	1.04	20.00	62.50	C	
Do	213	1.04	20.00	62.50	C	

[a] Including 13 grams of the original wrapper.

$P = 1.911A + 0.5B - 1.411C = 72.61$ kilograms per square centimeter.
$V = 15{,}000$ cubic centimeters. $S = 1.04$. $W = 213$ grams.
$M = \frac{VPS}{W} = 5{,}318$ kilograms per square centimeter (75,640 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 18, 1910.	Grams.
Solid	67.1
Liquid (water)	14.4
Gaseous	122.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.5 grams.[a]

Date (1912).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Dec. 13	81.95	0.794	669.0
Do	81.95	.776	673.2
Dec. 16	81.95	.773	670.6

[a] Including 6.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 677.6.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
June 2	63.50	52.25	11.25
Do	63.50	51.75	11.75
Do	63.50	51.50	12.00

Average compression, 11.7 millimeters (0.46 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	222	159	15
Do	63	218	155	15
Do	63	213	150	15

Average expansion of bore hole, 155 cubic centimeters (9.46 cubic inches).

TUNNELITE NO. 6-L. F.

Explosive, Tunnelite No. 6-L. F.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 214 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.02.
Color of explosive, ecru with green spots.
Consistency, fibrous, coarse, dry, hard, moderately cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, December 13, 1909.
Unit swing on this date, 3.14 inches.
Weight of charge, in grams, 305, 305, 305.
Swing, in inches, 3.23, 3.18, 3.13.
Average swing, in inches, 3.18.

3.18 : 3.14 : : 305 : (301).

Therefore the unit deflective charge of Tunnelite No. 6-L. F. is 301 grams.

GAS AND DUST GALLERY NO. 1.

Date(1909-10).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Dec. 21	301	7.78	No ignition.	Jan. 26	301	.....	No ignition.
Do	301	7.81	Do.	Do	301	.....	Do.
Do	301	7.81	Do.	Do	301	.....	Do.
Dec. 22	301	8.16	Do.	Do	301	.....	Do.
Do	301	8.16	Do.	Do	301	.....	Do.
Do	301	7.98	Do.	Do	301	.....	Do.
Do	301	8.16	Do.	Do	301	.....	Do.
Do	301	8.16	Do.	TEST 4.			
Do	301	8.16	Do.				
Do	301	8.07	Do.	Jan. 20	a 680	3.98	No ignition.
				Do	a 680	4.09	Do.
TEST 3.				Jan. 21	a 680	4.14	Do.
Jan. 26	301	.....	No ignition.	Do	a 680	3.90	Do.
Do	301	.....	Do.	Do	a 680	3.90	Do.
Do	301	.....	Do.				

a 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 15	14.92	43.0	2,882
Dec. 17	14.22	43.0	3,024
Do	14.86	43.0	2,894

Average rate of detonation, 2,933 meters (9,620 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 24	13.75	17.37	6.00	0.300
Do	12.50	15.79	5.50	.275
Do	16.75	21.16	8.25	.412

Average height of flame, 18.11 inches.
Average duration of flame, 0.329 millisecond.

Impact Test.

Date, June 29, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
18	1	No explosion.	27	1	Explosion.
20	1	Do.	26	1	Do.
24	1	Do.	25	5	No explosion.
28	1	Explosion.			

The maximum height at which no explosion occurs established at 25 centimeters (9.84 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 166 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 19	6	Did not explode.	Nov. 19	4	Did not explode.
Do	4	Do.	Do	4	Do.
Do	3	Exploded.			

The minimum distance at which no explosion occurs established at 4 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 27	218	1.02	26.75	66.88	A	
Do	218	1.02	26.00	65.00	A	65.83
Do	218	1.02	26.25	65.62	A	
Oct. 4	218	1.02	24.75	61.88	B	
Do	218	1.02	24.50	61.25	B	61.46
Do	218	1.02	24.50	61.25	B	
Do	218	1.02	23.75	59.38	C	
Do	218	1.02	24.25	60.62	C	60.21
Do	218	1.02	24.25	60.62	C	

[a] Including 18 grams of the original wrapper.

P=1.911A+0.5B−1.411C=71.57 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.02. W=218 grams.

$M=\frac{VPS}{W}$=5,023 kilograms per square centimeter (71,450 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, July 1, 1910.	Grams.
Solid	65.0
Liquid (water)	12.0
Gasecus	132.7

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 110.0 grams.[a]

Date (1910).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Aug. 23	80.95	0.759	630.5
Aug. 24	80.95	.746	619.5
Do	80.95	.736	611.1

[a] Including 10 grams of the original wrapper.

Average large calories per kilogram of explosive, 620.4.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
Mar. 12	63.50	52.75	10.75
Do	63.50	52.75	10.75
Do	63.50	53.50	10.00

Average compression, 10.5 millimeters (0.41 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Oct. 19	63	214	151	15
Do	63	210	147	15
Do	63	214	151	15

Average expansion of bore hole, 150 cubic centimeters (9.15 cubic inches).

TUNNELITE NO. 7.

Explosive, Tunnelite No. 7.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 200 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 0.98.
Color of explosive, écru with green spots.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, August 17, 1909.
Unit swing on this date, 2.75 inches.
Weight of charge, in grams, 305, 305, 305.
Swing, in inches, 2.64, 2.68, 2.77.
Average swing, in inches, 2.70.

2.70 : 2.75 :: 305 : (311).

Therefore the unit deflective charge of Tunnelite No. 7 is 311 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.	*Grams.*	*Per cent.*		TEST 3—Con.	*Grams.*	*Per cent.*	
Aug. 20	311	8.45	No ignition.	Aug. 25	311	.....	No ignition.
Do	311	8.62	Do.	Do	311	.....	Do.
Do	311	8.62	Do.	Do	311	.....	Do.
Do	311	8.62	Do.	Aug. 26	311	.....	Do.
Aug. 23	311	7.97	Do.	Do	311	.....	Do.
Do	311	8.32	Do.	Do	311	.....	Do.
Do	311	8.34	Do.	Do	311	.....	Do.
Do	311	8.20	Do.	TEST 4.			
Do	311	8.19	Do.				
Do	311	8.40	Do.	Aug. 16	[a] 680	4.22	No ignition.
				Do	[a] 680	4.22	Do.
TEST 3.				Do	[a] 680	4.22	Do.
				Do	[a] 680	4.14	Do.
Aug. 25	311	.....	No ignition.	Do	[a] 680	4.22	Do.
Do	311	.....	Do.				
Do	311	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 4	16.18	43.0	2,658
Do	16.17	43.0	2,659
Do	16.16	43.0	2,661

Average rate of detonation, 2,659 meters (8,720 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 12	20.50	25.89	10.00	0.500
Do	18.75	23.68	8.00	.400
Do	18.75	23.68	7.75	.388

Average height of flame, 24.42 inches.
Average duration of flame, 0.429 millisecond.

Impact Test.

Date, May 5, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
14	1	No explosion.	13	3	No explosion.
16	1	Explosion.	13	1	Explosion.
15	1	Do.	12	2	No explosion.
14	1	No explosion.	12	1	Explosion.
14	1	Explosion.	11	5	No explosion.

The maximum height at which no explosion occurs established at 11 centimeters (4.33 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 164 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 4	14	Did not explode.	Dec. 5	15	Did not explode.
Do	12	Do.	Do	15	Exploded.
Dec. 5	10	Exploded.	Do	16	Do.
Do	11	Do.	Do	17	Did not explode.
Do	12	Do.	Do	17	Do
Do	13	Do.	Do	17	Do.
Do	14	Do.			

The minimum distance at which no explosion occurs established at 17 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter = 1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 4	211	1.02	20.50	64.06	A	
Do	211	1.02	20.25	63.28	A	62.76
Do	211.	1.02	19.50	60.94	A	
Feb. 3	211	1.02	19.50	60.94	B	
Do	211	1.02	19.50	60.94	B	60.68
Do	211	1.02	19.25	60.16	B	
Do	211	1.02	18.00	56.25	C	
Do	211	1.02	18.25	57.03	C	57.29
Do	211	1.02	18.75	58.59	C	

[a] Including 11 grams of the original wrapper.

$P=1.911A+0.5B-1.411C=69.44$ kilograms per square centimeter.
$V=15{,}000$ cubic centimeters. $S=1.02$. $W=211$ grams.
$M=\frac{VPS}{W}=5{,}035$ kilograms per square centimeter (71,610 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 11 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, December 12, 1912.	Grams.
Solid	52.0
Liquid (water)	10.0
Gaseous	123.8

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 105.5 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 10	81.95	0.697	609.6
Do	81.95	.714	624.6
Jan. 11	81.95	.722	631.8

[a] Including 5.5 grams of the original wrapper.

Average large calories per kilogram of explosive, 622.0.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
June 2	63.50	53.50	10.00
Do	63.50	53.00	10.50
Do	63.50	51.75	11.75

Average compression, 10.8 millimeters (0.43 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	212	149	15
Do	63	209	146	15
Do	63	208	145	15

Average expansion of bore hole, 147 cubic centimeters (8.97 cubic inches).

TUNNELITE NO. 8.

Explosive, Tunnelite No. 8.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1¼ inches.
Length of cartridge, 8 inches.
Average weight, 209 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.00.
Color of explosive, ecru with green spots.
Consistency, granular, fine, dry, soft, slightly cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, August 17, 1909.
Unit swing on this date, 2.75 inches.
Weight of charge, grams, 310, 310, 310.
Swing, in inches, 2.60, 2.62, 2.67.
Average swing, in inches, 2.63.

2.63 : 2.75 : : 310 : (324).

Therefore the unit deflective charge of Tunnelite No. 8 is 324 grams.

GAS AND DUST GALLERY NO. 1.

Date (1909).	Weight of charge.	Methane and ethane.	Result.	Date (1909).	Weight of charge.	Methane and ethane.	Result.
TEST 1.				TEST 3—Con.			
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
Aug. 23	324	8.29	No ignition.	Aug. 24	324	.....	No ignition.
Do	324	8.37	Do.	Do	324	.....	Do.
Do	324	8.59	Do.	Do	324	.....	Do.
Do	324	8.36	Do.	Do	324	.....	Do.
Do	324	8.54	Do.	Do	324	.....	Do.
Aug. 24	324	8.02	Do.	Do	324	.....	Do.
Do	324	8.40	Do.	Do	324	.....	Do.
Do	324	8.26	Do.	TEST 4.			
Do	324	8.20	Do.				
Do	324	8.56	Do.	Aug. 13	[a] 680	4.16	No ignition.
				Aug. 14	[a] 680	4.25	Do.
TEST 3.				Do	[a] 680	4.25	Do.
				Do	[a] 680	4.16	Do.
Aug. 24	324	.....	No ignition.	Do	[a] 680	4.25	Do.
Do	324	.....	Do.				
Do	324	.....	Do.				

[a] 680 grams or more was used.

RATE OF DETONATION.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1910).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
June 6	17.23	43.0	2,496
Do	18.05	43.0	2,382
Do	17.49	43.0	2,459

Average rate of detonation, 2,446 meters (8,020 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Apr. 12	16.00	20.21	9.75	0.488
Do	15.50	19.58	6.50	.325
Do	21.75	27.47	11.75	.588

Average height of flame, 22.42 inches.
Average duration of flame, 0.467 millisecond.

Impact Test.

Date, May 6, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
12	1	No explosion.	11	1	Explosion.
14	1	Explosion.	10	1	No explosion.
13	1	Do.	10	1	Explosion.
12	1	Do.	9	1	Do.
11	2	No explosion.	8	5	No explosion.

The maximum height at which no explosion occurs established at 8 centimeters (3.15 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 172 grams.

Date (1912).	Distance separating cartridges.	Result, upper cartridge.	Date (1912).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Dec. 5	15	Exploded.	Dec. 5	18	Did not explode.
Do	17	Do.	Do	18	Do.
Do	18	Did not explode.			

The minimum distance at which no explosion occurs established at 18 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Date (1913).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
Feb. 4	212	1.07	20.25	63.28	A	
Do	212	1.07	20.00	62.50	A	62.50
Do	212	1.07	19.75	61.72	A	
Do	212	1.07	18.75	58.59	B	
Do	212	1.07	19.25	60.16	B	59.64
Feb. 13	212	1.07	19.25	60.16	B	
Feb. 4	212	1.07	18.50	57.81	C	
Feb. 5	212	1.07	19.25	60.16	C	59.12
Do	212	1.07	19.00	59.38	C	

[a] Including 12 grams of the original wrapper.

P=1.911A+0.5B−1.411C=65.84 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.07. W=212 grams.

$M=\frac{VPS}{W}$=4,985 kilograms per square centimeter (70,900 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 14 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst, A. L. Hyde.]

Date, June 17, 1910.	Grams.
Solid	78.5
Liquid (water)	16.0
Gaseous	116.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 106.0 grams.[a]

Date (1913).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Jan. 18	81.95	0.790	688.8
Do	81.95	.803	700.2
Jan. 20	81.95	.768	669.3

[a] Including 6 grams of the original wrapper.

Average large calories per kilogram of explosive, 686.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before explosion.	After explosion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
June 2	63.50	53.50	10.00
Do	63.50	53.25	10.25
Do	63.50	53.50	10.00

Average compression, 10.1 millimeters (0.40 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
May 26	63	188	125	15
Do	63	199	136	15
Do	63	199	136	15

Average expansion of bore hole, 132 cubic centimeters (8.05 cubic inches).

TUNNELITE NO. 8–L. F.

Explosive, Tunnelite No. 8–L. F.
Class 4, nitroglycerin.
Manufactured by the G. R. McAbee Powder & Oil Co.
Physical examination:
Diameter of cartridge, 1⅛ inches.
Length of cartridge, 8 inches.
Average weight, 218 grams.
Cartridge had not been redipped.
Apparent specific gravity of cartridge by sand, 1.05.
Color of explosive, ecru with green spots.
Consistency, granular and fibrous, coarse, dry, hard, moderately cohesive.
Unit deflective charge as determined by the ballistic pendulum:
Date, December 13, 1909.
Unit swing on this date, 3.14 inches.
Weight of charge, in grams, 320, 320, 320.
Swing, in inches, 3.18, 3.10, 3.08.
Average swing, in inches, 3.12.

3.12 : 3.14 : : 320 : (322).

Therefore the unit deflective charge of Tunnelite No. 8–L. F. is 322 grams.

Gas and Dust Gallery No. 1.

Date(1909–10).	Weight of charge.	Methane and ethane.	Result.	Date (1910).	Weight of charge.	Methane and ethane.	Result.
Test 1.	*Grams.*	*Per cent.*		Test 3—Con.	*Grams.*	*Per cent.*	
Dec. 21	322	7.98	No ignition.	Jan. 26	322	.....	No ignition.
Do	322	8.32	Do.	Jan. 27	322	.....	Do.
Do	322	8.32	Do.	Do	322	.....	Do.
Do	322	8.32	Do.	Do	322	.....	Do.
Do	322	8.16	Do.	Do	322	.....	Do.
Do	322	7.98	Do.	Do	322	.....	Do.
Do	322	7.81	Do.	Do	322	.....	Do.
Do	322	7.81	Do.				
Do	322	7.81	Do.	Test 4.			
Do	322	7.81	Do.	Jan. 20	[a] 680	4.23	No ignition.
				Do	[a] 680	4.14	Do.
Test 3.				Do	[a] 680	3.93	Do.
Jan. 26	322	.....	No ignition.	Do	[a] 680	3.93	Do.
Do	322	.....	Do.	Do	[a] 680	4.14	Do.
Do	322	.....	Do.				

[a] 680 grams or more was used.

Rate of Detonation.

Diameter of cartridge, 1¼ inches.
Electric detonator, No. 7.

Date (1909).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
	Millimeters.	*Meters.*	*Meters.*
Dec. 17	16.55	43.0	2,598
Dec. 18	15.96	43.0	2,694
Do	16.03	43.0	2,682

Average rate of detonation, 2,658 meters (8,720 feet) per second.

Flame Test.

Peripheral speed of film, 20 meters per second.

Date (1910).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
	Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
Sept. 26	11.00	13.89	5.00	0.250
Sept. 29	9.75	12.32	5.75	.288
Do	13.00	16.42	6.00	.300

Average height of flame, 14.21 inches.

Average duration of flame, 0.279 millisecond.

Impact Test.

Date, June 29, 1910.

Distance of fall.	Number of trials.	Result.	Distance of fall.	Number of trials.	Result.
Centimeters.			*Centimeters.*		
18	1	No explosion.	17	1	Explosion.
22	1	Explosion.	16	1	Do.
20	1	Do.	15	1	No explosion.
19	1	Do.	15	1	Explosion.
18	2	No explosion.	14	1	Do.
18	1	Explosion.	13	5	No explosion.
17	1	No explosion.			

The maximum height at which no explosion occurs established at 13 centimeters (5.12 inches).

Explosion-by-Influence Test.

Weight of each cartridge, 169 grams.

Date (1910).	Distance separating cartridges.	Result, upper cartridge.	Date (1910).	Distance separating cartridges.	Result, upper cartridge.
	Inches.			*Inches.*	
Nov. 20	3	Exploded.	Nov. 21	7	Did not explode.
Nov. 21	4	Do.	Do	7	Do.
Do	6	Do.	Do	7	Do.
Do	8	Did not explode.			

The minimum distance at which no explosion occurs established at 7 inches.

Theoretical Maximum Pressure Developed in Own Volume, as Determined by Bichel Pressure Gage.

Indicator spring, 0.4 millimeter=1 kilogram per square centimeter.

Date (1910).	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
	Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
June 28	218	1.05	26.75	64.38	A	
Do	218	1.05	25.00	62.50	A	62.92
Do	218	1.05	24.75	61.88	A	
Oct. 5	218	1.05	23.25	58.12	B	
Do	218	1.05	23.75	59.38	B	58.75
Do	218	1.05	23.50	58.75	B	
Oct. 6	218	1.05	23.50	58.75	C	
Do	218	1.05	22.75	56.88	C	57.29
Do	218	1.05	22.50	56.25	C	

[a] Including 18 grams of the original wrapper.

P=1.911A+0.5B−1.411C=68.78 kilograms per square centimeter.
V=15,000 cubic centimeters. S=1.05. W=218 grams.
$M=\frac{VPS}{W}$=4,969 kilograms per square centimeter (70,680 pounds per square inch).

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF THE EXPLOSIVE AND 18 GRAMS OF THE ORIGINAL WRAPPER.

[Analyst. A. L. Hyde.]

Date, July 6, 1910.	Grams.
Solid	73.6
Liquid (water)	15.0
Gaseous	129.4

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Charge, 108.4 grams.[a]

Date (1910).	Weight of water.	Rise in temperature.	Heat developed per kilogram.
	Kilograms.	*°C.*	*Calories.*
Aug. 24	80.95	0.742	625.3
Aug. 25	80.95	.757	638.0
Do	80.95	.758	638.9

[a] Including 8.4 grams of the original wrapper.

Average large calories per kilogram of explosive, 634.1.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Date (1910).	Height.		Compression.
	Before expansion.	After expansion.	
	Millimeters.	*Millimeters.*	*Millimeters.*
July 9	63.50	54.25	9.25
Do	63.50	53.75	9.75
Do	63.50	54.00	9.50

Average compression, 9.5 millimeters (0.37 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Date (1910).	Volume of bore hole.		Expansion of bore hole.	Temperature of block.
	Before shot.	After shot.		
	Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*	*°C.*
Oct. 19	63	200	137	15
Do	63	200	137	15
Do	63	200	137	15

Average expansion of bore hole, 137 cubic centimeters (8.36 cubic inches).

RESULTS OF TESTS WITH SIX FOREIGN EXPLOSIVES.

The methods followed at all stations that are testing coal-mining explosives were devised so as to classify explosives with respect to their suitability for use in coal mines. The testing galleries in which explosives are tested vary in size, form, and material. Steel cannon in which the charges of explosives are fired are used at all stations except Austria, the bore of the cannon varying from 40 to 55 millimeters.

Natural gas is used at some stations and artificial substitutes at others. The limit charge established for a given explosive is by no means the same at each station; moreover, in some cases there is even a marked dissimilarity of results.

The use of stemming at foreign stations and at the Pittsburgh experiment station has been found to raise the limit charge considerably, especially of the ammonium nitrate class of explosives.

Air-spacing inside the bore of the cannon, as practiced at most of the Continental stations, is said to extend the apparent border line of safety. However, it may be stated that the results obtained with different charging densities at the Pittsburgh experiment station do not indicate that the border line of safety is necessarily extended with low-charging densities; on the other hand, some explosives have passed the test with a charge of 680 grams when loaded with a charging density of 1 that would not pass with this amount when loaded air spaced.

It has been shown by some of the German stations that some of the permitted explosives were less safe when tested in the presence of coal dust alone. The results obtained at the German stations were not always confirmed by the other stations where pure artificial or natural gas is used. This may be explained by the fact that the pit gases used at many of the stations are not of uniform composition and contain more or less inert gases.

There is sometimes a small percentage of carbon dioxide and other inert gases present in the pit gases used abroad, depending on the efficiency and condition of the scrubbers at the time the gases are drawn through. Assuming that the percentage of these inert gases is kept within low limits, there may be at times sufficient inert gases present to have a retardative effect on the ignition. Tests made at the Pittsburgh experiment station indicate that a gas and air mixture containing 8.1 per cent of methane and ethane, 14.2 per cent of carbon dioxide (CO_2), and 77.7 per cent of air is practically non-explosive when ignition is attempted with an electric igniter containing 5 grains of black powder. It was also found that the ignition of gas mixtures containing 11.7 per cent carbon dioxide is materially retarded and, even when ignited, burned at a slow rate accompanied

with little if any explosion. For this reason it would be fair to assume that a gas mixture containing a small percentage of carbon dioxide or other inert gases would be proportionately less sensitive than a pure gas mixture.

The results of tests of the four foreign explosives reported in Bulletin 15 indicated that the natural gas used at the Pittsburgh experiment station is more sensitive than the pit gas used abroad, and that it would be necessary to reduce the percentage of gas in order that the results might be comparable to those obtained abroad. It was evident that gas and air mixtures containing about 6 per cent of methane and ethane were equivalent to the 8 per cent mixtures used abroad, and further that all explosives would be excluded if untamped shots were fired in the presence of air mixtures containing 8 per cent of methane and ethane.

A conference of several of the leading coal operators was called at the Pittsburgh experiment station, and the consensus of opinion was that a low percentage gas mixture with coal dust suspended in the air would more nearly represent the dangerous conditions that might exist in a coal mine, and that one of the tests, at least, should be made under these conditions with unstemmed shots, using a charge of not less than 680 grams (1½ pounds).

STANDARDIZING APPARATUS AND METHODS.

The question of standardizing universal methods and apparatus for testing coal-mining explosives is still an open one. It is obvious that an interrelation of the work of all stations aiming to unify or standardize apparatus and methods of testing explosives would be beneficial to all concerned, and it is hoped that such will eventually be the outcome.

It would seem that the design in the construction of all testing apparatus could be established on a uniform basis. In fact, it is the tendency at present to adopt the cylindrical type of steel gallery and steel cannon having a bore of 55 millimeters in diameter. The unifying of test requirements does not offer a ready solution. In Belgium the mines are unusually fiery and all explosives are required to pass a severe gas test. In recent years explosives have been tested in the presence of coal dust, but the dust used has a relatively low percentage of volatile matter as compared with that used in the United States.

In France great stress is put on the requirements of the gaseous products of combustion of explosives. The nitroglycerin class of explosives is practically eliminated from use in French coal mines. When permissible explosives of this class are properly used in the coal mines of the United States in quantities less than 1½ pounds per shot,

it has been found that the amount of poisonous gases produced is so small that it is of little, if any, consequence. This is especially true when the ventilation current is brought directly to the working face. A testing gallery has recently been installed in France for determining the permissibility of explosives, hence the classification of explosives by calculating the theoretical temperature of explosion will no doubt be discontinued.

In England stemmed charges only are used for determining the permissibility of explosives for use in coal mines. This requirement would not always meet the conditions of mining in the United States. Although it is a reprehensible practice, miners have been known to fire permissible explosives in a drill hole with little, if any, stemming. For this reason unstemmed shots are one of the requirements of the Bureau of Mines for determining the permissibility of an explosive. It is understood that the new English testing station which is now in course of construction will test explosives without stemming in the presence of inflammable gas mixtures.

Six foreign coal-mining explosives were procured for comparative tests. The method of conducting the tests was approximately the same as that employed at the foreign testing stations.

The results of tests indicate that all of the foreign explosives, except explosive H, which detonated incompletely, would pass the Woolwich tests, but that all would fail to pass the Belgian tests. Only two of the explosives, namely, I and J, would pass the test requirements requisite for their being placed on the United States permissible list. In the gas and dust gallery tests explosive E failed in test 4; explosive F was not tested, as it failed to detonate completely in Belgian test; explosive G failed in test 4; and explosive H failed in test 1. Two of these explosives, namely, E and F, also failed in friction test.

The foreign explosives, namely, E, G, I, and J, were subjected to a suspended test, which simulates to some extent the procedure followed at one of the foreign testing stations. The test was made by suspending in the gallery 454 grams (1 pound) of the explosive, including the original wrapper. The charge was detonated in a mixture of gas and air containing 4 per cent of gas (methane and ethane) and 20 pounds of bituminous coal dust, 18 pounds of which was placed on shelves along the sides of the first 20 feet of the gallery, and 2 pounds placed so that it was stirred up by an air current in such a manner that all or a part of it would be suspended in the first division of the gallery. The explosive I was the only explosive that passed this test. The explosives G and J ignited the mixture in the first trial and explosive E ignited the mixture once out of two trials.

For convenience in reporting the results of tests, the six foreign explosives are here designated explosives E, F, G, H, I, and J.

Explosive E.
Class, potassium chlorate.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 12.7 centimeters (5 inches).
Average weight, 249 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.28.
Color of explosive, yellow.
Consistency, powdered, very fine, dry, soft, slightly cohesive.

Explosive F.
Class, potassium chlorate.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 14.6 centimeters (5¾ inches).
Average weight, 233 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.04.
Color of explosive, corn.
Consistency, powdered, very fine, dry, soft, not cohesive.

Explosive G.
Class, ammonium nitrate, containing nitroglycerin.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 17.1 centimeters (6¾ inches).
Average weight, 237 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.93.
Color of explosive, corn.
Consistency, granular and fibrous, fine, dry, soft, slightly cohesive.

Explosive H.
Class, ammonium nitrate.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 15.5 centimeters (6¼ inches).
Average weight, 242 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 1.03.
Color of explosive, corn.
Consistency, granular, fine, dry, hard, slightly cohesive.

Explosive I.
Class, ammonium nitrate.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 18.4 centimeters (7¼ inches).
Average weight, 250 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.86.
Color of explosive, corn.
Consistency, powdered, fine, dry, soft, not cohesive.

Explosive J.
Class, ammonium nitrate.
Physical examination:
Diameter of cartridge, 4.45 centimeters (1¾ inches).
Length of cartridge, 15.2 centimeters (6 inches).
Average weight, 226 grams.
Cartridge had been redipped.
Apparent specific gravity of cartridge by sand, 0.94.
Color of explosive, corn.
Consistency, powdered, fine, dry, soft, not cohesive.

Chemical Analyses.

Chemical analyses of these six samples resulted as follows:

EXPLOSIVE E.

[Analyst, A. L. Hyde.]

Moisture	0.17
Dinitrotoluene	17.85
Mononitronaphthalene	1.30
Castor oil	5.32
Potassium chlorate	75.36
	100.00

Ash, 0.04.

EXPLOSIVE F.

Moisture	0.10
Nitrated rosin	24.63
Potassium chlorate	75.27
	100.00

Ash, 0.03.

EXPLOSIVE G.

Moisture	0.52
Nitroglycerin	8.13
Nitrotoluene	3.57
Castor oil	.81
Nitrocellulose	.56
Ammonium nitrate	82.11
Wood pulp	4.30
	100.00

Ash, 0.04.

EXPLOSIVE H.

Moisture	0.18
Ammonium nitrate	88.17
Trinitrotoluene	8.13
Mononitronaphthalene	3.52
	100.00

Ash, 0.07.

EXPLOSIVE I.

[Analyst, C. G. Storm.]

Moisture	0.06
Rosin	5.01
Ammonium nitrate	91.43
Potassium nitrate	3.50
	100.00

Ash, 0.08.

EXPLOSIVE J.

Moisture	0.45
Trinitrotoluene	4.82
Ammonium nitrate	90.50
Flour	4.23
	100.00

Ash, 0.07.

PRODUCTS OF COMBUSTION FROM 200 GRAMS OF EACH EXPLOSIVE.

[Analyst, A. L. Hyde.]

Explosive.	Date (1912).	Test No.	Grams of original wrapper.	Products of combustion.		
				Gaseous.	Solid.	Liquid (water).
				Grams.	*Grams.*	*Grams.*
E	June 24	P 877	10	96.8	90.2	10.5
F	June 25	P 878	6	92.5	94.2	7.2
G	July 1	P 882	13	121.5	.0	70.0
H	July 2	P 883	10	120.8	1.2	71.0
I	July 3	P 884	10	110.5	3.1	80.5
J	July 6	P 886	8	106.2	2.1	81.0

GASEOUS PRODUCTS OF COMBUSTION, PERCENTAGE BY VOLUME.

Explosive.	Date (1912).	Test No.	H_2S.	CO_2.	CO.	O_2.	H_2.	CH_4.	N_2.
			Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
E	June 24	P 877	0.0	34.1	41.5	0.0	15.6	0.5	8.3
F	June 25	P 878	.0	23.4	47.4	.0	23.9	.6	4.7
G	July 1	P 882	.0	28.7	5.9	.0	11.0	.8	53.6
H	July 2	P 883	.0	27.4	5.5	.0	9.1	.7	57.3
I	July 3	P 884	.0	25.5	3.0	.0	9.6	1.1	60.8
J	July 6	P 886	.0	30.0	.0	.1	.0	.2	69.7

SOLID PRODUCTS OF COMBUSTION, PERCENTAGE BY WEIGHT.

Explosive.	Date (1912).	Test No.	Products of combustion.	
			Soluble.	Insoluble.
			Per cent.	*Per cent.*
E	June 24	P 877	90.43	9.57
F	June 25	P 878	89.60	10.40

LARGE CALORIES DEVELOPED BY 1 KILOGRAM OF THE EXPLOSIVE.

Explosive.	Date (1912).	Charge.[a]	Test No.	Weight of water.	Rise in temperature.	Heat developed per kilogram.	Average heat developed per kilogram.
		Grams.		*Kilograms*	*°C.*	*Calories.*	*Calories.*
E	July 10	105	C 580	81.95	1.234	1,090.7	1,065.1
	11	105	C 581	81.95	1.176	1,039.0	
	11	105	C 582	81.95	1.206	1,065.7	
F	13	103	C 585	81.95	.966	868.5	865.5
	13	103	C 586	81.95	.957	860.4	
	15	103	C 587	81.95	.965	867.7	
G	15	106.5	C 588	81.95	1.300	1,133.2	1,169.2
	16	106.5	C 591	81.95	1.363	1,188.5	
	17	106.5	C 592	81.95	1.360	1,185.9	
H	17	105.0	C 593	(b)	(b)	(b)	.....
	17	105.0	C 594	(b)	(b)	(b)	
I	17	105.0	C 595	81.95	1.371	1,212.7	1,218.6
	18	105.0	C 596	81.95	1.408	1,245.6	
	18	105.0	C 597	81.95	1.354	1,197.5	
J	19	104.0	C 598	81.95	1.439	1,285.5	.....
	19	104.0	C 599	[c] 81.95	1.221	1,089.4	
	19	104.0	C 600	[c] 81.95	1.240	1,106.5	

[a] Charge consists of 100 grams of the explosive ingredient plus the proportionate amount of the original wrapper.
[b] Owing to erratic results obtained, results of tests not reported, incomplete detonation, nitrogen oxides evolved.
[c] Incomplete detonation—nitrogen oxides evolved.

COMPRESSION OF SMALL LEAD BLOCKS.

Charge, 100 grams.

Explosive.	Date (1912).	Test No.	Height before explosion.	Height after explosion.	Compression.
			Millimeters.	*Millimeters.*	*Millimeters.*
E	June 18	B 855	63.50	45.50	18.00
	18	B 856	63.25	45.25	18.00
	18	B 857	63.25	45.00	18.25
F	18	B 858	63.25	43.75	19.50
	18	B 859	63.50	44.25	19.25
	18	B 860	63.25	44.00	19.25
G	18	B 862	63.50	44.00	19.50
	18	B 863	63.25	44.50	18.75
	18	B 864	63.25	44.00	19.25
H	18	B 865	63.25	46.75	16.50
	18	B 868	63.25	46.00	17.25
	18	B 869	63.25	46.00	17.25
I	18	B 870	63.25	44.25	19.00
	18	B 871	63.25	44.00	19.25
	18	B 872	63.50	44.25	19.25
J	18	B 873	63.25	44.00	19.25
	18	B 874	63.25	44.00	19.25
	18	B 875	63.25	43.50	19.75

The average compression is as follows:

Explosive E, 18.1 millimeters (0.71 inch).
Explosive F, 19.3 millimeters (0.76 inch).
Explosive G, 19.2 millimeters (0.76 inch).
Explosive H, 17.0 millimeters (0.67 inch).
Explosive I, 19.2 millimeters (0.76 inch).
Explosive J, 19.4 millimeters (0.76 inch).

EXPANSION OF BORE HOLE OF TRAUZL LEAD BLOCKS.

Charge, 10 grams.

Explosive.	Date (1912).	Test No.	Volume of bore hole before shot.	Volume of bore hole after shot.	Expansion of bore hole.
			Cubic centimeters.	*Cubic centimeters.*	*Cubic centimeters.*
E	July 20	A 887	62	278	216
	20	A 888	62	268	206
	20	A 889	62	277	215
F	20	A 892	62	260	198
	Aug. 1	A 893	62	261	199
	1	A 894	62	269	207
G	July 20	A 881	62	357	295
	20	A 882	62	362	300
	20	A 883	62	353	291
H	June 15	A 873	62	329	267
	15	A 874	62	338	276
	15	A 875	62	343	281
I	15	A 877	62	288	226
	15	A 878	62	296	234
	15	A 876	62	274	[a] 212
J	July 20	A 884	62	303	241
	20	A 885	62	303	241
	20	A 886	62	315	253

[a] In this and other tests explosive detonated incompletely.

The average expansions follow:

Explosive E, 212 cubic centimeters (12.93 cubic inches).
Explosive F, 201 cubic centimeters (12.26 cubic inches).
Explosive G, 295 cubic centimeters (18.00 cubic inches).
Explosive H, 275 cubic centimeters (16.78 cubic inches).
Explosive J, 245 cubic centimeters (14.94 cubic inches).

UNIT DEFLECTIVE CHARGE AS DETERMINED BY THE BALLISTIC PENDULUM.

Explosive.	Date (1912).	Charge.	Swing.	Average swing.	Unit deflective charge.
		Grams.	*Inches.*	*Inches.*	*Grams.*
Pittsburgh experiment station 40 per cent standard dynamite	June 18	227	3.37		
	18	227	3.30	3.32	227
	18	227	3.28		
E	18	240	3.47		
	18	240	3.43	3.44	232
	18	240	3.41		
F	18	250	3.38		
	18	250	3.38	3.36	247
	18	250	3.33		
Pittsburgh experiment station 40 per cent standard dynamite	17	227	3.38		
	17	227	3.31	3.35	227
	17	227	3.35		
G	17	200	3.32		
	17	200	3.27	3.29	204
	17	200	3.28		
H	17	210	3.29		
	17	210	3.35	3.32	212
	17	210	3.31		
I	17	225	3.32		
	17	225	3.38	3.37	224
	17	225	3.42		
J	17	230	3.21		
	17	230	3.16	3.20	241
	17	230	3.23		

In order to test the foreign explosives at the Pittsburgh experiment station under conditions simulating the procedure followed at Woolwich, England, it was necessary to determine the standard Woolwich charge of each explosive. This was computed from the results of ballistic pendulum tests of the four foreign explosives reported in Bulletin 15, Chapter VIII. It was found that the Woolwich disruptive charge for an explosive was 69 per cent of that used at the Pittsburgh experiment station. The following tabulation gives the unit deflective charge and the Woolwich disruptive charge of the four foreign explosives as determined at the Pittsburgh experiment station:

Explosive.	Pittsburgh experiment station unit deflective charge.	Woolwich disruptive charge.	Percentage Woolwich charge compared with charge used at the Pittsburgh experiment station.
	Grams.	*Grams.*	
A	420	295	70
B	205	144	70
C	202	137	68
D	298	199	67
Average			69

Therefore, the Woolwich disruptive charge is 69 per cent of the charge used at the Pittsburgh experiment station, and this factor (0.69) was used in computing the standard Woolwich charge of the six foreign explosives reported herein.

In all tests in gas and dust gallery No. 1 the following electric detonators were used:

Explosive E, electric detonator No. 6.
Explosive F, electric detonator No. 6.
Explosive G, electric detonator No. 6.
Explosive H, electric detonator No. 7.
Explosive I, electric detonator No. 7.
Explosive J, electric detonator No. 7.

THE BELGIAN TEST (MODIFIED) FOR DETERMINATION OF THE LIMIT CHARGE IN A MIXTURE OF GAS AND AIR CONTAINING 8 PER CENT OF METHANE AND ETHANE.

Limit charge is established if five shots fail to ignite the mixture.

Explosive.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
		Grams.	*Per cent.*	
E	June 27	50	7.99	No ignition.
	27	50	8.17	Do.
	28	50	8.09	Do.
	28	50	7.94	Do.
	28	50	7.66	Ignition.
F	27	100	7.91	No ignition.[a]
	27	100	8.06	Do.[a]
	27	250	8.50	Do.[a]
G	July 1	50	8.03	Ignition.
H	June 28	50	7.78	No ignition.
	28	50	7.55	Ignition.
I	29	50	8.02	Do.
J	26	50	8.23	Do.

[a] Detonation incomplete.

The limit charges for this test are established as follows:

	Grams.
Explosive E	0
Explosive G	0
Explosive H	0
Explosive I	0
Explosive J	0

Woolwich Test (Modified).

Five shots with the standard (Woolwich) charge, in its original wrapper, shall be fired with 12 inches of clay stemming (tamped by hand) at a gallery temperature of 77° F. into a mixture of gas and air containing 8 per cent of methane and ethane. An explosive passes this test if all five shots fail to ignite the mixture.

Explosive.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
		Grams.	*Per cent.*	
E	Aug. 21	160	8.06	No ignition.
	21	160	8.05	Do.
	21	160	7.82	Do.
	21	160	8.46	Do.
	21	160	7.92	Do.
F	21	170	8.11	Do.
	21	170	7.90	Do.
	21	170	7.88	Do.
	21	170	7.88	Do.
	21	170	8.24	Do.
G	22	141	8.54	Do.
	23	141	8.40	Do.
	23	141	8.25	Do.
	23	141	8.07	Do.
	23	141	7.78	Do.
H	21	146	8.11	Do.[a]
	22	146	8.25	Do.[a]
	22	146	7.83	Do.[a]
I	22	155	8.00	Do.
	22	155	8.25	Do.
	22	155	8.11	Do.
	22	155	8.39	Do.
	22	155	8.05	Do.
J	22	166	8.31	Do.
	22	166	7.83	Do.
	22	166	7.93	Do.
	23	166	8.01	Do.
	23	166	7.98	Do.

[a] Detonation incomplete.

Explosives E, F, G, I, and J passed this test.

WOOLWICH TEST (MODIFIED).

Five shots with three-quarters of the standard (Woolwich) charge, in its original wrapper, shall be fired with 9 inches of clay stemming (tamped by hand) at a gallery temperature of 77° F. into a mixture of gas and air containing 8 per cent of methane and ethane. An explosive passes this test if all five shots fail to ignite the mixture.

Explosive.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
		Grams.	*Per cent.*	
E	Aug. 26	120	8.25	No ignition.
	26	120	7.93	Do.
	26	120	7.99	Do.
	26	120	7.83	Do.
	26	120	8.11	Do.
F	24	128	7.84	Do.
	24	128	7.77	Do.
	24	128	8.01	Do.
	24	128	8.03	Do.
	24	128	8.23	Do.
G	23	106	7.79	Do.
	23	106	7.96	Do.
	24	106	8.38	Do.
	24	106	7.82	Do.
	24	106	7.90	Do.
H (this test was not made on this explosive, as it failed to detonate completely in previous test with 12 inches of stemming).				
I	26	116	7.75	Do.
	26	116	8.50	Do.
	26	116	8.20	Do.
	26	116	8.44	Do.
	26	116	8.04	Do.
J	23	124	7.83	Do.
	23	124	7.79	Do.
	23	124	8.11	Do.
	23	124	7.79	Do.
	23	124	7.80	Do.

Explosives E, F, G, I, and J passed this test.

BUREAU OF MINES TESTS.

GAS AND DUST GALLERY No. 1.

Test 1.

Explosive.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
		Grams.	*Per cent.*	
E	June 19	232	8.39	No ignition.
	19	232	8.23	Do.
	19	232	8.30	Do.
	19	232	8.29	Do.
	19	232	8.27	Do.
	19	232	8.31	Do.
	19	232	8.06	Do.
	20	232	7.91	Do.
	20	232	7.94	Do.
	20	232	8.33	Do.
F	20	247	7.88	Do.
	20	247	7.94	Do.
	20	247	8.30	Do.
	20	247	8.43	Do.
	20	247	8.33	Do.
	20	247	8.30	Do.
	20	247	7.87	Do.
	20	247	8.29	Do.
	20	247	8.45	Do.
	20	247	7.99	Do.
G	24	204	8.40	Do.
	24	204	8.14	Do.
	24	204	8.15	Do.
	24	204	8.12	Do.
	24	204	8.19	Do.
	24	204	8.39	Do.
	24	204	8.12	Do.
	24	204	8.08	Do.
	24	204	7.90	Do.
	24	204	8.33	Do.
H	24	212	7.92	Do.
	24	212	8.14	Do.
	25	212	8.07	Ignition.
I	21	224	8.01	No ignition.
	21	224	8.12	Do.
	22	224	8.36	Do.
	22	224	8.16	Do.
	22	224	8.04	Do.
	22	224	8.31	Do.
	22	224	8.06	Do.
	22	224	8.30	Do.
	22	224	7.98	Do.
	22	224	8.28	Do.
J	20	241	8.02	Do.
	21	241	8.06	Do.
	21	241	8.05	Do.
	21	241	8.20	Do.
	21	241	7.90	Do.
	21	241	8.26	Do.
	21	241	8.36	Do.
	21	241	8.06	Do.
	21	241	8.01	Do.
	21	241	8.34	Do.

Explosives E, F, G, I, and J passed this test.
Explosive H failed to pass this test.

GAS AND DUST GALLERY No. 1.

Test 3.

Explosive.	Date (1912).	Weight of charge.	Result.
		Grams.	
E	Aug. 16	232	No ignition.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
	16	232	Do.
F	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
	20	247	Do.
G	16	204	Do.
	16	204	Do.
	16	204	Do.
	16	204	Do.
	16	204	Do.
	17	204	Do.
	17	204	Do.
	17	204	Do.
	17	204	Do.
	17	204	Do.
H	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
	20	212	Do.
I	17	224	Do.
	17	224	Do.
	17	224	Do.
	17	224	Do.
	17	224	Do.
	19	224	Do.
	19	224	Do.
	19	224	Do.
	19	224	Do.
	19	224	Do.
J	19	241	Do.
	19	241	Do.
	19	241	Do.
	19	241	Do.
	19	241	Do.
	19	241	Do.
	19	241	Do.
	19	241	Do.
	20	241	Do.
	20	241	Do.

Explosives E, F, G, H, I, and J passed this test.

GAS AND DUST GALLERY NO. 1.

Test 4.

Explosive.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
		Grams.	*Per cent.*	
E	Aug. 8	500	4.13	Ignition.
	8	350	4.18	No ignition.
	8	400	4.12	Do.
	9	450	4.13	Do.
	9	450	4.10	Ignition.
	9	400	3.92	No ignition.
	9	400	4.13	Ignition.
	9	350	4.25	Do.
	9	300	4.21	No ignition.
	9	300	4.15	Do.
	9	300	4.17	Do.
	9	300	4.19	Do.
	9	300	4.14	Do.
F (this test was not run on this explosive, as it failed to detonate completely in previous test when stemming was not used).				
G	2	100	3.89	Do.
	2	150	3.96	Do.
	3	200	4.42	Do.
	3	250	3.83	Do.
	3	350	3.92	Do.
	3	450	3.85	Ignition.
	3	400	4.15	No ignition.
	3	400	4.00	Ignition.
	5	200	4.14	No ignition.
	5	350	4.08	Do.
	5	350	4.10	Do.
	5	350	4.04	Ignition.
	5	300	4.03	No ignition.
	6	300	4.10	Do.
	6	300	4.23	Do.
	6	300	4.20	Do.
	9	300	4.16	Do.
H (this test not run, as explosive failed in Test 1).				
I	14	900	4.34	Do.
	14	1,000	4.25	Do.
J	8	350	4.03	Do.
	8	450	4.20	Do.
	8	600	4.22	Do.
	8	700	4.26	Do.
	8	800	4.05	Do.
	10	900	4.08	Do.
	10	1,000	4.21	Do.

For this test the following limit charges were established:

	Grams.
Explosive E	300
Explosive G	300
Explosive I (the capacity of the cannon)	1,000
Explosive J (the capacity of the cannon)	1,000

The following limit charges were established when the explosives were loaded air spaced. The diameter of the bore of the cannon was 2¼ inches (57 mm.) and the explosives were packed in cartridges having a diameter of 1¾ inches (45 mm.), except for explosive I, which was packed in a 2-inch (51-mm.) cartridge.

	Grams.
Explosive E	250
Explosive G	50
Explosive I	800
Explosive J	300

Rate of Detonation.

Diameter of cartridge, 1¼ inches.[a]
Electric detonator, No. 7.

Explosive.	Date (1912).	Distance between spark points.	Peripheral speed of drum per second.	Rate of detonation per second.
		Millimeters.	*Meters.*	*Meters.*
E[b]	June 27	15.70	43.5	2,771
F[b]	27	16.35	44.5	2,722
G	24	11.35	44.5	3,921
	25	11.60	45.0	3,879
	25	11.20	45.0	4,018
H	26	10.80	45.0	4,167
	26	10.30	45.0	4,369
	26	10.35	45.0	4,348
I	21	10.60	45.0	4,245
	21	10.70	45.0	4,206
	21	10.70	45.0	4,206
J	25	11.15	44.5	3,991
	26	11.50	45.0	3,913
	26	11.10	44.5	4,009

[a] The explosives H and I failed to detonate completely in 1¼-inch cartridges, therefore 1¼-inch cartridges were used.
[b] Explosive placed in iron pipe 1½ inches inside diameter and 42 inches long and closed at both ends by means of iron plugs.

The average rates of detonation of the explosives are as follows:
Explosive G, 3,939 meters per second (12,920 feet per second).
Explosive H, 4,295 meters per second (14,090 feet per second).
Explosive I, 4,219 meters per second (13,840 feet per second).
Explosive J, 3,971 meters per second (13,020 feet per second).

Flame Test.

Peripheral speed of film, 20 meters per second.

Explosive.	Date (1912).	Height of photograph.	Height of flame.	Duration distance.	Duration of flame.
		Millimeters.	*Inches.*	*Millimeters.*	*Milliseconds.*
E	July 2	9.75	12.32	7.00	0.350
	2	6.75	8.53	2.25	.112
	2	7.25	9.16	2.50	.125
F	2	10.00	12.63	7.25	.362
	2	9.75	12.32	5.75	.288
	2	9.50	12.00	5.50	.275
G	3	9.00	11.37	6.50	.325
	3	7.50	9.47	5.25	.262
	3	7.50	9.47	5.00	.250
H	3	(a)	(a)	(a)	(a)
I	5	7.75	9.79	2.25	.112
	5	7.00	8.84	2.50	.125
	9	7.25	9.16	1.75	.088
J	5	6.75	8.53	5.00	.250
	5	7.00	8.84	5.75	.288
	5	7.25	9.16	6.00	.300

[a] Incomplete detonation.

The average heights and durations of flame are as follows:

Explosive E, height 10.00 inches; duration, 0.196 millisecond.
Explosive F, height 12.32 inches; duration 0.308 millisecond.
Explosive G, height 10.10 inches; duration 0.279 millisecond.
Explosive I, height 9.26 inches; duration 0.108 millisecond.
Explosive J, height 8.84 inches; duration 0.279 millisecond.

SMALL IMPACT TEST.

Explosive.	Date (1912).	Distance of fall.	Number of trials.	Result.
		Centimeters.		
E	June 21	30	1	Explosion.
	21	10	1	No explosion.
	21	20	1	Do.
	21	25	1	Explosion.
	21	24	1	Do.
	21	23	5	No explosion.
F	21	25	1	Explosion.
	21	20	1	Do.
	21	10	1	Do.
	21	8	1	Do.
	21	6	1	Do.
	21	5	2	No explosion.
	21	5	1	Explosion.
	21	4	1	No explosion.
	21	4	1	Explosion.
	21	3	5	No explosion.
G	July 8	40	1	Explosion.
	8	36	1	No explosion.
	8	38	1	Do.
	8	39	5	Do.
H	9	60	1	Explosion.
	9	56	1	No explosion.
	9	58	1	Explosion.
	9	57	5	No explosion.
I	June 28	40	1	Do.
	28	60	1	Do.
	28	80	1	Do.
	28	100	1	Explosion.
	28	96	1	No explosion.
	28	98	1	Do.
	28	99	1	Explosion.
	28	98	1	Do.
	28	97	1	Do.
	July 8	96	1	Do.
	8	90	1	No explosion.
	8	93	1	Do.
	8	95	1	Explosion.
	8	94	1	Do.
	8	93	1	Do.
	8	92	1	Do.
	8	91	1	No explosion.
	8	91	1	Explosion.
	8	90	1	No explosion.
	8	90	1	Explosion.
	8	89	1	No explosion.
	8	89	1	Explosion.
	8	88	1	Do.
	8	87	1	No explosion.
	8	87	1	Explosion.
	8	86	1	Do.
	8	85	1	Do.
	8	84	2	No explosion.
	8	84	1	Explosion.
	8	83	1	Do.
	8	82	1	Do.
	8	81	1	Do.
	8	80	2	No explosion.
	8	80	1	Explosion.
	8	79	1	No explosion.
	8	79	1	Explosion.
	8	78	1	Do.
	8	77	1	No explosion.
	8	77	1	Explosion.
	8	76	5	No explosion.
J	9	60	1	Do.
	9	80	1	Do.
	9	100	5	Do.

The maximum heights that will cause no explosion established as follows:

Explosive E, 23 centimeters (9.06 inches).
Explosive F, 3 centimeters (1.18 inches).
Explosive G, 39 centimeters (15.35 inches).
Explosive H, 57 centimeters (22.44 inches).
Explosive I, 76 centimeters (29.92 inches).
Explosive J, 100 centimeters (39.37 inches), capacity of machine.

LARGE IMPACT TEST.

Explosive.	Date (1912).	Distance of fall.	Number of trials.	Amount recovered.	Result.
		Centimeters.		*Grams.*	
E	July 1	150	1	.....	Explosion.
	1	60	1	.....	Do.
	1	20	1	18	Do.
	1	10	3	20	No explosion.
F	May 31	100	1	.....	Explosion.
	31	30	1	2	Do.
	31	20	1	1	Do.
	31	10	3	20	No explosion.
G	June 8	100	1	10	Explosion.
	8	50	1	10	Do.
	8	20	1	20	No explosion.
	8	40	3	20	Do.
H	17	100	1	20	Do.
	17	200	1	9	Explosion.
	17	160	1	20	No explosion.
	17	180	1	10	Explosion.
	17	170	1	10	Do.
	17	160	2	20	No explosion.
I	18	500	1	.....	Explosion.
	18	300	1	8	Do.
	18	200	1	9	Do.
	18	100	1	20	No explosion.
	18	120	1	12	Explosion.
	18	110	3	20	No explosion.
J	19	600	1	.....	Explosion.
	19	400	1	8	Do.
	19	200	1	9	Do.
	19	160	1	20	No explosion.
	19	190	3	20	Do.

The maximum heights that will cause no explosion established as follows:

Explosive E, 10 centimeters (3.94 inches).
Explosive F, 10 centimeters (3.94 inches).
Explosive G, 40 centimeters (15.75 inches).
Explosive H, 160 centimeters (62.99 inches).
Explosive I, 110 centimeters (43.31 inches).
Explosive J, 190 centimeters (74.80 inches).

EXPLOSION-BY-INFLUENCE TEST.

Explosive.	Date (1912).	Weight of each cartridge.	Distance separating cartridges.	Result, upper cartridge.
		Grams.	*Inches.*	
E	Aug. 26	206	4	Did not explode.
	26	206	1	Exploded.
	26	206	3	Did not explode.
	27	206	2	Exploded
	27	206	3	Did not explode.
	27	206	3	Do.
F	27	167	.....	Incomplete detonation.
G	27	150	3	Did not explode.
	28	150	1	Exploded.
	28	150	2	Do.
	28	150	3	Did not explode.
	28	150	3	Do.
H (this test not made, as explosive failed to detonate completely in flame test).				
I	29	138	3	Do.
	29	138	1	Exploded.
	29	138	2	Did not explode.
	29	138	2	Do.
	29	138	2	Do.
J	28	151	3	Do.
	28	151	1	Exploded.
	28	151	2	Did not explode.
	28	151	2	Exploded.
	28	151	3	Did not explode.
	28	151	3	Do.

The minimum distance at which no explosion occurs established as follows:

	Inches.
Explosive E	3
Explosive G	3
Explosive I	2
Explosive J	3

THEORETICAL MAXIMUM PRESSURE DEVELOPED IN OWN VOLUME, AS DETERMINED BY BICHEL PRESSURE GAGE.

Indicator spring, 0.32 millimeter=1 kilogram per square centimeter.

Explosive.	Date (1912).	Test No.	Charge.[a]	Specific gravity.	Height of curve.	Pressure per square centimeter.	Cooling surface.	Average pressure per square centimeter.
			Grams.		*Millimeters.*	*Kilograms.*		*Kilograms.*
E	July 8	P 887	210	1.28	26.00	81.25	A	79.95
	8	P 888	210	1.28	26.00	81.25	A	
	8	P 891	210	1.28	24.75	77.34	A	
	8	PP 1137	210	1.28	24.25	75.78	B	
	8	PP 1139	210	1.28	23.50	73.44	B	74.74
	9	PP 1141	210	1.28	24.00	75.00	B	
	9	PP 1142	210	1.28	23.50	73.44	C	
	9	PP 1143	210	1.28	23.00	71.88	C	72.66
	9	PP 1144	210	1.28	23.25	72.66	C	
F	10	P 892	206	1.04	23.00	71.88	A	
	10	P 894	206	1.04	23.00	71.88	A	71.88
	10	P 896	206	1.04	23.00	71.88	A	
	10	PP 1149	206	1.04	22.25	69.53	B	
	10	PP 1151	206	1.04	21.75	67.97	B	68.23
	10	PP 1152	206	1.04	21.50	67.19	B	
	9	PP 1146	206	1.04	21.00	65.62	C	
	9	PP 1147	206	1.04	20.50	64.06	C	64.06
	9	PP 1148	206	1.04	20.00	62.50	C	
G	11	P 897	213	.93	38.75	121.09	A	
	11	P 898	213	.93	38.75	121.09	A	120.57
	11	P 899	213	.93	38.25	119.53	A	
	11	PP 1153	213	.93	35.50	110.94	B	
	11	PP 1154	213	.93	35.50	110.94	B	110.94
	11	PP 1155	213	.93	35.50	110.94	B	
	11	PP 1156	213	.93	34.75	108.59	C	
	11	PP 1157	213	.93	34.50	107.81	C	107.81
	12	PP 1158	213	.93	34.25	107.03	C	
H	12	P 900	210	1.03	36.75	114.84	A	
	13	P 902	210	1.03	37.00	115.62	A	115.36
	13	P 903	210	1.03	37.00	115.62	A	
	12	PP 1162	210	1.03	34.50	107.81	B	
	12	PP 1164	210	1.03	34.50	107.81	B	108.07
	13	PP 1165	210	1.03	34.75	108.59	B	
	12	PP 1159	210	1.03	33.00	103.12	C	
	12	PP 1160	210	1.03	33.50	104.69	C	104.95
	12	PP 1161	210	1.03	34.25	107.03	C	
I	19	P 910	210	.86	37.50	117.19	A	
	20	P 911	210	.86	38.00	118.75	A	116.67
	20	P 912	210	.86	36.50	114.06	A	
	16	PP 1166	210	.86	32.75	102.34	B	
	16	PP 1167	210	.86	32.75	102.34	B	103.12
	16	PP 1168	210	.86	33.50	104.69	B	
	16	PP 1169	210	.86	32.25	100.78	C	
	16	PP 1170	210	.86	31.25	97.66	C	100.00
	16	PP 1171	210	.86	32.50	101.56	C	
J	18	P 907	208	.94	36.00	112.50	A	
	18	P 908	208	.94	37.00	115.62	A	114.84
	18	P 909	208	.94	37.25	116.41	A	
	17	PP 1175	208	.94	32.25	100.78	B	
	17	PP 1176	208	.94	33.50	104.69	B	102.60
	17	PP 1177	208	.94	32.75	102.34	B	
	17	PP 1172	208	.94	32.00	100.00	C	
	17	PP 1173	208	.94	32.00	100.00	C	100.52
	17	PP 1174	208	.94	32.50	101.56	C	

[a] Charge consists of 200 grams of explosive ingredient plus the proportionate amount of the original wrapper.

Computations for the respective explosives, established as the result of the above tests, were as follows:

Explosive E: P=1.911A+0.5B−1.411C=87.63 kilograms per square centimeter. V=15,000 cubic centimeters. S=1.28. W=210 grams. $M=\frac{V \cdot PS}{W}$=8,012 kilograms per square centimeter (113,960 pounds per square inch).

Explosive F: P=1.911A+0.5B−1.411C=81.09 kilograms per square centimeter. V=15,000 cubic centimeters. S=1.04. W=206 grams. $M=\frac{VPS}{W}$=6,141 kilograms per square centimeter (87,340 pounds per square inch).

Explosive G: P=1.911A+0.5B−1.411C=133.76 kilograms per square centimeter. V=15,000 cubic centimeters. S=0.93. W=213 grams. $M=\frac{VPS}{W}$=8,760 kilograms per square centimeter (124,600 pounds per square inch).

Explosive H: P=1.911A+0.5B−1.411C=126.40 kilograms per square centimeter. V=15,000 cubic centimeters. S=1.03. W=210 grams. $M=\frac{VPS}{W}$=9,299 kilograms per square centimeter (132,260 pounds per square inch).

Explosive I: P=1.911A+0.5B−1.411C=133.42 kilograms per square centimeter. V=15,000 cubic centimeters. S=0.86. W=210 grams. $M=\frac{VPS}{W}$=8,196 kilograms per square centimeter (116,570 pounds per square inch).

Explosive J: P=1.911A+0.5B−1.411C=128.93 kilograms per square centimeter. V=15,000 cubic centimeters. S=0.94. W=208 grams. $M=\frac{VPS}{W}$=8,740 kilograms per square centimeter (124,310 pounds per square inch).

PENDULUM FRICTION TEST.

Explosive.	Date (1912).	Test No.	Charge on grooved steel plate.	Weights used.	Distance of fall.	Kind of shoe.	Result.
			Grams.	*Kilograms.*	*Meters.*		
E.....	Mar. 14	U 211	7	20	1½	Steel....	Explosion in first of 10 trials (180 swings).
E.....	Apr. 4	U 222	7	20	1½	Fiber...	No explosion or burning in 10 trials (180 swings), but local cracking in second and fourth trials.[a]
F.....	Mar. 14	U 212	7	20	1½	Steel....	Explosion in fourth, fifth, seventh, and ninth of 10 trials (180 swings).
F.....	Apr. 4	U 223	7	20	1½	Fiber...	Explosion in eighth of 10 trials (180 swings).
G.....	...do....	U 218	7	20	1½	Steel....	No explosion or burning in 10 trials (180 swings), but local cracking in third and ninth trials.[a]
H.....	...do....	U 219	7	20	1½	Steel....	No explosion or burning in 10 trials (180 swings), but local cracking in seventh trial.[a]
I.....	...do....	U 220	7	20	1½	Steel....	No explosion or burning in 10 trials (180 swings), but local cracking in second, fifth, eighth, and ninth trials.[a]
J.....	...do....	U 221	7	20	1½	Steel....	No explosion or burning in 10 trials (180 swings), but local cracking in first, fourth, seventh, and tenth trials.[a]

[a] Scarcely audible.

Explosives E and F failed to pass this test.

Friction tests were made with the six foreign explosives under conditions simulating those prevailing in England. Approximately 10 grams of each explosive was spread ⅛ inch thick on a wooden block and struck 10 glancing blows with a rawhide mallet. Sharp cracking occurred with explosives E and F but no audible cracking was noted when the other explosives were tested. It is worthy of note that in no instance did a complete explosion occur, but when tested with the pendulum friction machine explosives E and F detonated completely.

LIMIT CHARGE WITH VARIOUS GAS AND AIR MIXTURES.

The following tabulations show the limit charges established with the foreign explosives E, G, I, and J when fired in the presence of various gas and air mixtures. No stemming was used in any of the tests, but tests were made with different loading densities in the bore of the cannon.

LIMIT CHARGE TESTS WITH EXPLOSIVE E.

(Charging density of 1.)

Test in gas mixture containing 5 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 26	400	5.14	No ignition.	July 29	700	5.06	No ignition.
Do	400	5.11	Do.	Do	750	5.00	Ignition.
Do	500	5.13	Do.	July 30	700	4.97	No ignition.
Do	600	5.22	Do.	Do	700	5.01	Ignition.
Do	700	5.23	Do.	Do	600	4.94	No ignition.
Do	800	5.03	Do.	Aug. 1	650	5.22	Ignition.
July 27	900	5.11	Do.	Do	600	5.07	No ignition.
Do	1,000	5.04	Ignition.	Do	600	5.01	Do.
Do	900	5.16	Do.	Do	600	4.91	Do.
July 29	800	5.19	Do.				

Limit charge established at 600 grams for explosive E when fired with a charging density of 1 in the presence of a 5 per cent gas mixture.

Test in gas mixture containing 6 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 1	50	6.04	No ignition.	July 16	200	6.02	Ignition.
July 10	200	5.97	Do.	Do	150	6.00	No ignition.
Do	400	6.17	Ignition.	Do	150	6.06	Do.
July 12	200	5.95	No ignition.	July 17	150	6.28	Do.
July 13	250	6.18	Do.	Do	150	6.25	Do.
July 15	300	6.25	Do.	Do	150	6.14	Do.
July 16	350	5.93	Do.				

Limit charge established at 150 grams for explosive E when fired with a charging density of 1 in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 28.....	100	6.99	Ignition.	July 2	50	7.21	No ignition.
July 1.....	50	6.94	No ignition.	Do.....	50	6.92	Do.
July 2.....	50	6.82	Do.	Do.....	50	6.97	Do.

Limit charge established at 50 grams for explosive E when fired with a charging density of 1 in the presence of a 7 per cent gas mixture.

Test in gas mixture containing 8 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 27.....	50	7.99	No ignition.	June 28	50	7.94	No ignition.
Do.....	50	8.17	Do.	Do.....	50	7.66	Ignition.
June 28.....	50	8.09	Do.				

Limit charge established at 0 gram for explosive E when fired with a charging density of 1 in the presence of an 8 per cent gas mixture.

Test in gas mixture containing 8½ per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 27.....	200	8.25	Ignition.	June 27	50	8.45	No ignition.
Do.....	150	8.37	Do.	Do.....	50	8.52	Do.
Do.....	100	8.85	Do.	June 28.....	50	7.66	Ignition.

Limit charge established at 0 gram for explosive E when fired with a charging density of 1 in the presence of an 8½ per cent gas mixture.

Test in gas mixture containing 9 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 27.....	51	8.80	No ignition.	July 18.....	50	8.99	Ignition.

Limit charge established at 0 gram for explosive E when fired with a charging density of 1 in the presence of a 9 per cent gas mixture.

Test in gas mixture containing 10 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 20.....	100	9.83	Ignition.	July 22.....	50	10.14	No ignition.
July 18.....	50	9.86	No ignition.	July 23.....	50	9.93	Do.
Do.....	50	10.09	Do.	Do.....	50	10.14	Do.

Limit charge established at 50 grams for explosive E when fired with a charging density of 1 in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE G.

(Charging density of 1.)

Test in gas mixture containing 5 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 24.....	200	5.19	Ignition.	July 25.....	50	5.23	No ignition.
Do.....	100	4.89	Do.	Do.....	50	4.84	Do.
July 25.....	50	4.97	No ignition.	Do.....	50	5.09	Do.
Do.....	50	5.10	Do.				

Limit charge established at 50 grams for explosive G when fired with a charging density of 1 in the presence of a 5 per cent gas mixture.

On July 1, 1912, a 50-gram charge was fired into a gas mixture containing 6.23 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 11.....	50	6.87	No ignition.	July 11.....	50	7.03	Ignition.
Do.....	100	6.76	Ignition.				

Limit charge established at 0 gram for explosive G when fired with a charging density of 1 in the presence of a 7 per cent gas mixture.

On July 1, 1912, a 50-gram charge was fired into a gas mixture containing 8.03 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of an 8 per cent gas mixture.

Test in gas mixture containing 8½ per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
June 26.....	100	8.51	Ignition.	June 26.....	50	8.42	Ignition.

Limit charge established at 0 gram for explosive G when fired with a charging density of 1 in the presence of an 8½ per cent gas mixture.

On July 17, 1912, a 50-gram charge was fired into a gas mixture containing 9.17 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 9 per cent gas mixture.

On July 18, 1912, a 50-gram charge was fired into a gas mixture containing 10.16 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE I.

(Charging density of 1.)

Test in gas mixture containing 6 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 8.....	50	6.22	No ignition.	July 9.....	400	5.99	No ignition.
Do.....	100	5.95	Do.	Do.....	500	6.16	Do.
Do.....	150	6.30	Do.	Do.....	600	6.26	Do.
Do.....	150	6.28	Do.	Do.....	700	5.47	Do.
Do.....	200	5.99	Do.	July 23.....	800	5.75	Do.
July 9.....	250	6.29	Do.	July 24.....	900	6.04	Do.
Do.....	300	6.13	Do.	Do.....	1,000	6.06	Do.
Do.....	350	6.35	Do.				

Limit charge established at 1,000 grams (the capacity of the cannon) for explosive I when fired with a charging density of 1 in the presence of a 6 per cent gas mixture.

On July 3, 1912, a 50-gram charge was fired into a gas mixture containing 7.20 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 7 per cent gas mixture.

On June 29, 1912, a 50-gram charge was fired into a gas mixture containing 8.02 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of an 8 per cent gas mixture.

On June 25, 1912, a 50-gram charge was fired into a gas mixture containing 8.72 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of an 8½ per cent gas mixture.

On July 18, 1912, a 50-gram charge was fired into a gas mixture containing 8.80 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 9 per cent gas mixture.

Test in gas mixture containing 10 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 22.....	50	10.25	No ignition.[a]	July 23.....	50	9.86	No ignition.[a]
July 23.....	50	10.30	Do.[a]				

[a] Incomplete detonation; tests discontinued.

Limit charge established at 0 gram for explosive I when fired with a charging density of 1 in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE J.

(Charging density of 1.)

Test in gas mixture containing 5 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 26.....	200	4.85	No ignition.	July 27.....	700	5.17	No ignition.
Do.....	300	4.83	Do.	Do.....	900	5.20	Do.
Do.....	400	4.92	Do.	July 29.....	1,000	5.10	Do.
Do.....	500	5.04	Do.				

Limit charge established at 1,000 grams (the capacity of the cannon) for explosive J when fired with a charging density of 1 in the presence of a 5 per cent gas mixture.

On July 6, 1912, a 50-gram charge was fired into a gas mixture containing 6.24 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 5.....	50	7.05	No ignition.	July 5.....	50	6.94	Ignition.
Do.....	100	7.09	Ignition.				

Limit charge established at 0 gram for explosive J when fired with a charging density of 1 in the presence of a 7 per cent gas mixture.

On June 26, 1912, a 50-gram charge was fired into a gas mixture containing 8.23 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of an 8 per cent gas mixture.

Test in gas mixture containing 9 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 17.....	50	9.21	No ignition.	July 17.....	50	9.80	Ignition.

Limit charge established at 0 gram for explosive J when fired with a charging density of 1 in the presence of a 9 per cent gas mixture.

On July 18, 1912, a 50-gram charge was fired into a gas mixture containing 9.97 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired with a charging density of 1 in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE E.

(Charge air spaced.)

Test in gas mixture containing 5 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 30.....	650	5.08	Ignition.	July 31.....	450	4.85	Ignition.
Do.....	600	4.89	Do.	Do.....	400	5.17	No ignition.
Do.....	550	4.89	Do.	Do.....	400	5.14	Do.
July 31.....	400	5.13	No ignition.	Do.....	400	5.12	Do.
Do.....	450	5.15	Do.	Do.....	400	5.03	Do.
Do.....	450	5.09	Do.				

Limit charge established at 400 grams for explosive E when fired air spaced in the presence of a 5 per cent mixture.

Test in gas mixture containing 6 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 10.....	400	6.04	Ingition.	July 12.....	250	5.92	Ignition.
Do.....	250	6.14	No ignition.	July 13.....	200	6.02	No ignition.
July 11.....	300	6.14	Do.	Do.....	200	6.23	Do.
Do.....	350	6.13	Do.	Do.....	200	6.09	Ignition.
Do.....	350	6.21	Do.	Do.....	150	6.14	No ignition.
July 12.....	350	6.11	Do.	Do.....	150	6.86	Do.
Do.....	350	6.08	Ignition.	July 15.....	150	6.22	Do.
Do.....	250	5.91	No ignition.	Do.....	150	6.27	Do.
Do.....	250	6.16	Do.	July 17.....	150	6.07	Do.
Do.....	300	6.09	Ignition.				

Limit charge established at 150 grams for explosive E when fired air spaced in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 2.....	100	6.75	No ignition.	July 2.....	100	7.13	No ignition.
Do.....	100	6.83	Do.	Do.....	100	7.09	Do.
Do.....	150	6.68	Ignition.	Do.....	100	7.11	Do.

Limit charge established at 100 grams for explosive E when fired air spaced in the presence of a 7 per cent gas mixture.

On June 28, 1912, a 50-gram charge was fired into a gas mixture containing 8.10 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8 per cent gas mixture.

Test in gas mixture containing 9 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams*	*Per cent.*	
July 18.....	50	9.00	No ignition.	July 18.....	50	8.96	Ignition.

Limit charge established at 0 gram for explosive E when fired air spaced in the presence of a 9 per cent gas mixture.

Test in gas mixture containing 10 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 19.....	150	9.63	Ignition.	July 22.....	50	10.08	No ignition.
July 20.....	100	9.42	Do.	Do.....	50	10.19	Do.
Do.....	50	9.94	No ignition.	Do.....	50	10.03	Do.
July 22.....	100	10.22	Ignition.	Do.....	50	10.21	Do.

Limit charge established at 50 grams for explosive E when fired air spaced in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE G.

(Charge air spaced.)

Test in gas mixture containing 5 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 25.....	100	4.92	No ignition.	July 25.....	50	5.04	No ignition.
Do.....	150	4.85	Do.	Do.....	50	5.19	Do.
Do.....	200	5.10	Ignition.	Do.....	50	4.90	Do.
Do.....	150	5.28	Do.	Do.....	50	5.23	Do.
Do.....	100	5.00	No ignition.	Do.....	50	5.11	Do.
Do.....	100	5.14	Ignition.				

Limit charge established at 50 grams for explosive G when fired air spaced in the presence of a 5 per cent gas mixture.

On July 1, 1912, a 50-gram charge was fired into a gas mixture containing 6.00 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 6 per cent gas mixture.

On July 11, 1912, a 50-gram charge was fired into a gas mixture containing 6.81 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 7 per cent gas mixture.

On July 1, 1912, a 50-gram charge was fired into a gas mixture containing 7.54 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8 per cent gas mixture.

On June 25, 1912, a 50-gram charge was fired into a gas mixture containing 8.43 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8½ per cent gas mixture.

On July 17, 1912, a 50-gram charge was fired into a gas mixture containing 8.97 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 9 per cent gas mixture.

On July 23, 1912, a 50-gram charge was fired into a gas mixture containing 9.99 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 10 per cent gas mixture.

LIMIT CHARGE TESTS WITH EXPLOSIVE I.

(Charge air spaced.)

On July 30, 1912, a 1,000-gram charge was fired into a gas mixture containing 5.95 per cent methane and ethane. Result: No ignition. The limit charge was established at 1,000 grams (the capacity of the cannon) when fired air spaced in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 3.....	50	6.81	No ignition.	July 5.....	150	7.00	No ignition.
Do.....	50	7.14	Do.	Do.....	150	7.05	Do.
Do.....	100	7.02	Do.	Do.....	150	7.23	Do.
Do.....	200	6.61	Ignition.	Do.....	150	7.04	Do.
Do.....	150	6.92	No ignition.				

Limit charge established at 150 grams for explosive I when fired air spaced in the presence of a 7 per cent gas mixture.

Test in gas mixture containing 8 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*	
July 1	50	7.83	No ignition.
Do	50	8.04	Ignition.

Limit charge established at 0 gram for explosive I when fired air spaced in the presence of an 8 per cent gas mixture.

On June 26, 1912, a 50-gram charge was fired into a gas mixture containing 8.58 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8½ per cent gas mixture.

On July 18, 1912, a 50-gram charge was fired into a gas mixture containing 8.93 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 9 per cent gas mixture.

Test in gas mixture containing 10 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*	
July 22	150	9.91	Ignition.
July 23	50	10.00	Do.

Limit charge established at 0 gram for explosive I when fired air spaced in the presence of a 10 per cent gas mixture.

Limit Charge Tests with Explosive J.

(Charge air spaced.)

On July 30, 1912, a 1,000-gram charge was fired into a gas mixture containing 4.88 per cent methane and ethane. Result: No ignition. The limit charge was established at 1,000 grams (the capacity of the cannon) when fired air spaced in the presence of a 5 per cent gas mixture.

Test in gas mixture containing 6 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.	Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*			*Grams.*	*Per cent.*	
July 6	100	6.15	Ignition.	July 8	50	6.21	No ignition.
July 8	50	6.07	No ignition.	Do	50	6.31	Do.
Do	50	5.95	Do.	Do	50	6.14	Do.

Limit charge established at 50 grams for explosive J when fired air spaced in the presence of a 6 per cent gas mixture.

Test in gas mixture containing 7 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*	
July 6	100	6.81	Ignition.
Do	50	6.90	No ignition.
Do	50	6.79	Ignition.

Limit charge established at 0 gram for explosive J when fired air spaced in the presence of a 7 per cent gas mixture.

On August 26, 1912, a 50-gram charge was fired into a gas mixture containing 7.90 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8 per cent gas mixture.

On June 26, 1912, a 50-gram charge was fired into a gas mixture containing 8.55 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of an 8½ per cent gas mixture.

On July 17, 1912, a 50-gram charge was fired into a gas mixture containing 9.01 per cent methane and ethane. Result: Ignition. The limit charge was established at 0 gram when fired air spaced in the presence of a 9 per cent gas mixture.

Test in gas mixture containing 10 per cent gas.

Date (1912).	Weight of charge.	Methane and ethane.	Result.
	Grams.	*Per cent.*	
July 18	50	9.72	Ignition.
July 23	50	10.55	Do.

Limit charge established at 0 gram for explosive J when fired air spaced in the presence of a 10 per cent gas mixture.

It is to be noted that the limit charge established with all of the foreign explosives tested in the presence of 8, 8½, and 9 per cent gas mixtures is 0 gram, even though the explosive was loaded with different charging densities (figs. 3, 4, 5, 6).

When these explosives were fired in the presence of 10 per cent gas the highest limit charge established was 50 grams (explosive E). The conditions of loading in the cannon were apparently not a factor in the limit charge established.

When the explosives were fired in the presence of 7 per cent gas the limit charge established was 0 gram for explosives G and J. The limit charge established for explosive I was 0 gram when loaded with a density of 1, and 150 grams when loaded air spaced. Explosive E also gave a higher limit charge when loaded air spaced than when loaded with a density of 1, namely, 100 and 50 grams, respectively.

When the explosives were fired in the presence of 6 per cent gas the limit charge established for explosive G was 0 gram, and 1,000 grams for explosive I, the results being the same irrespective of the charging

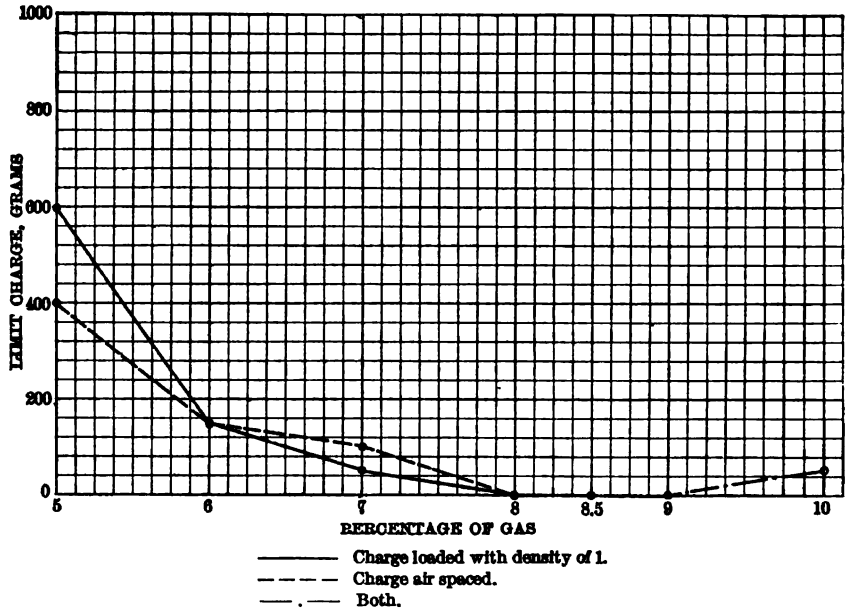


FIGURE 3.—Foreign explosive E. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.17; dinitrotoluene, 17.85; potassium chlorate, 75.36; castor oil, 5.32; mononitronaphthalene, 1.30; total 100.

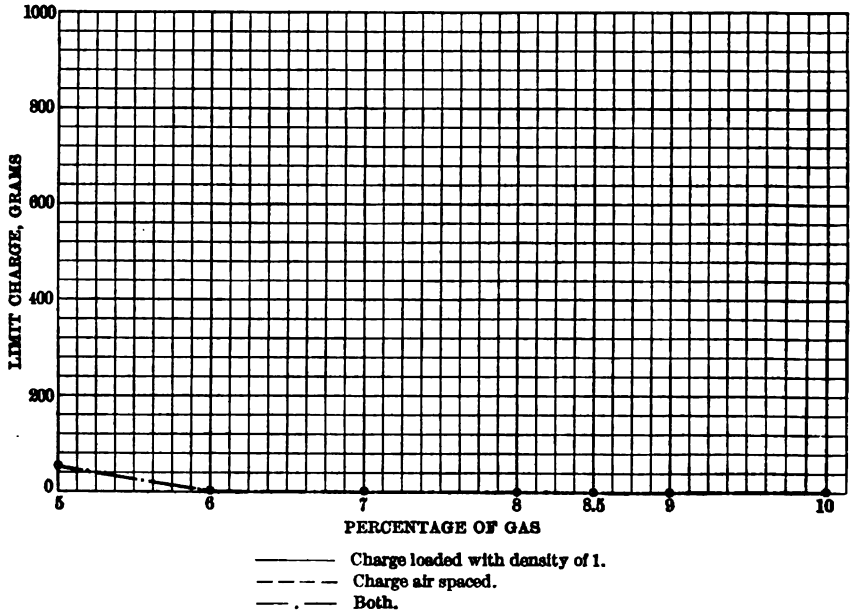


FIGURE 4.—Foreign explosive G. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.52; nitroglycerin, 8.13; ammonium nitrate, 82.11; nitrotoluene, 3.57; wood pulp, 4.30; castor oil, 0.81; nitrocellulose, 0.56; total, 100.

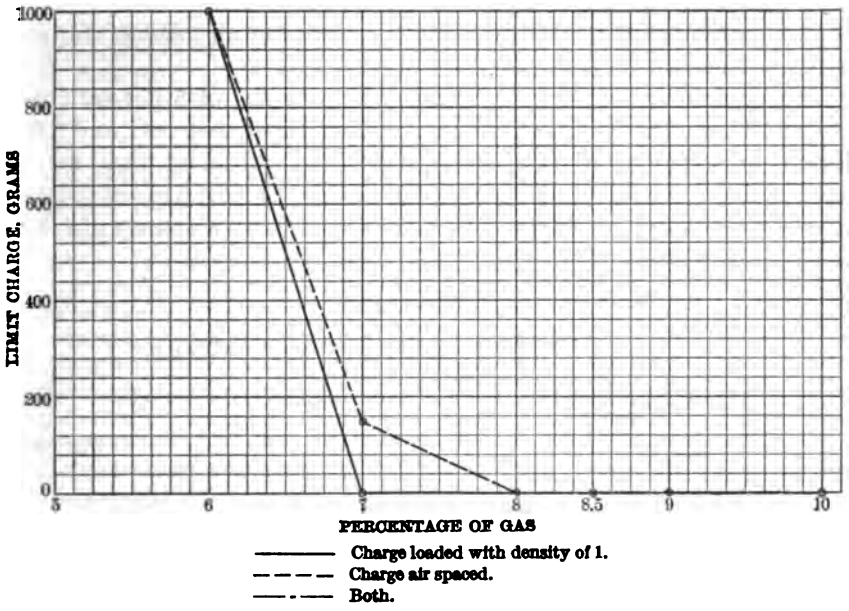


FIGURE 5.—Foreign explosive I. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.06; ammonium nitrate, 91.43; potassium nitrate, 3.50; rosin, 5.01; total, 100.

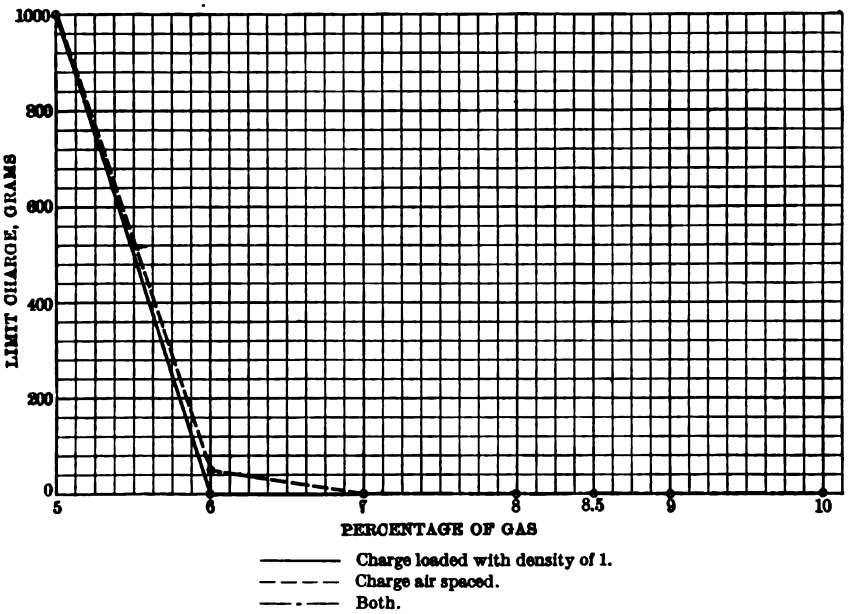


FIGURE 6.—Foreign explosive J. Limit charge established with various gas percentages. Composition of explosive: Moisture, 0.45; ammonium nitrate, 90.50; trinitrotoluene, 4.82; flour, 4.23; total, 100.

density. Explosive J gave a limit charge of 50 grams when air spaced, and 0 gram when loaded with a density of 1. The limit charge established for explosive E was 150 grams when loaded air spaced and also when loaded with a charging density of 1.

When the explosives were fired in the presence of 5 per cent gas, 50 grams was established as the limit charge for explosive G, and 1,000 grams for explosive J. The same results were obtained when loaded with different charging densities. Six hundred grams was established as the limit charge for explosive E when loaded with a density of 1, and 400 grams was established when loaded air spaced.

APPENDIX.

BUREAU OF MINES TESTS OF EXPLOSIVES.

CONDITIONS UNDER WHICH EXPLOSIVES ARE TESTED.

The conditions under which the Bureau of Mines tests explosives to determine whether they shall be placed on its list of permissible explosives are as follows:

1. The manufacturer is to deliver to the Bureau of Mines, Fortieth and Butler Streets, Pittsburgh, Pa., three weeks prior to the date set for tests, 100 pounds of each explosive that he desires to have tested. He is to be responsible for the care, handling, and delivery of this material to the experiment station, and he is to have a representative present during the tests if he so desires. In order to avoid duplication of work, it is requested that the smallest size of cartridge that the manufacturer intends to place on the market be sent for these tests.

2. The manufacturer will be required to pay fees for tests of explosives as outlined in Schedule 1[a] of the Bureau of Mines. This schedule may be procured from the Director of the Bureau of Mines, Washington, D. C.

3. No one is to be present at or participate in these tests except the necessary Government officers at the experiment station, their assistants, and the representative of the manufacturer of the explosives to be tested.

4. The tests will be made in the order of the receipt of the applications for them, provided the necessary quantity of the explosive is delivered at the testing station by the date set, of which date due notice will be given by the Bureau of Mines.

5. A list of the explosives that pass certain requirements satisfactorily will be furnished to State mine inspectors and will be made public in such other manner as may be considered desirable.

6. The details of results of tests are to be considered confidential by the manufacturer and are not to be made public prior to official publication by the Bureau of Mines.

7. From time to time field samples of permissible explosives will be collected, and tests will be made of these explosives as they are supplied for use in coal mines in the various States. Field samples when collected in original containers will not only be retested for their permissibility, but the analysis of each ingredient of the explosive must agree, within a reasonable variation, to the results obtained with the original sample submitted for tests.

TEST REQUIREMENTS FOR PERMISSIBLE EXPLOSIVES.

The tests will be made by the engineers of the Bureau of Mines in gas and dust gallery No. 1 at Pittsburgh, Pa., or in one of its identical galleries. The charge of explosive to be fired in tests 1 and 3 shall be equal in deflective power, as determined

[a] Fees for testing explosives and conditions and requirements under which explosives are tested. 1911. 6 pp.

by the ballistic pendulum, to one-half pound (227 grams) of 40 per cent nitroglycerin dynamite in its original wrapper, of the following formula:

	Per cent.
Nitroglycerin	40
Nitrate of soda (sodium nitrate)	44
Wood pulp	15
Carbonate of lime (calcium carbonate)	1
	100

Each charge shall be fired with an electric detonator (exploder or cap) which will completely detonate or explode the charge, as recommended by the manufacturer. The explosive must be in such condition that the chemical and physical tests do not show any unfavorable results.

An explosive will be considered unsatisfactory if it is not chemically stable, if it shows leakage of nitroglycerin, or if it is in such a condition that exudation of nitroglycerin will occur in handling or transportation.

A 680-gram (1½-pound) charge of an explosive must not evolve 158 liters (5½ cubic feet) or more of poisonous gases as determined by gage tests. The poisonous gases that are likely to be evolved on explosion are carbon monoxide, hydrogen sulphide, and nitrogen oxides.

Explosives that require a charge of more than 680 grams (1½ pounds) to satisfactorily break down coal under normal conditions of mining will be considered unsatisfactory. The size of the charge is determined by tests with the ballistic pendulum.

An explosive will be considered unsatisfactory if it is too sensitive to frictional impact (a glancing blow). Such sensitiveness is determined by the pendulum friction device when the shoe is faced with fiber.

In order that the dust used in tests 3 and 4 may be of the same quality, it is always taken from the same mine, ground to the same fineness, and used while still fresh.

The following are the gallery tests to which are subjected the explosives that the Bureau of Mines is asked to place in the list of permissible explosives:

Test 1. Ten shots each with the charge as described above, in its original wrapper, shall be fired, each tamped with 1 pound [a] of clay stemming, at a gallery temperature of 77° F., into a mixture of gas and air containing 8 per cent of gas (methane and ethane). An explosive is considered to have passed the test if no one of the 10 shots ignites this mixture.

Test 3. Ten shots each with the charge as described above, in its original wrapper, shall be fired, each tamped with 1 pound [a] of clay stemming, at a gallery temperature of 77° F., into 40 pounds of bituminous coal dust, 20 pounds of which is to be distributed uniformly on a wooden bench placed in front of the cannon and 20 pounds placed on side shelves in sections 4, 5, and 6. An explosive is considered to have passed the test if no one of the 10 shots ignites this mixture.

Test 4. Five shots each with 1½-pound charge, in its original wrapper, shall be fired without stemming, at a gallery temperature of 77°F., into a mixture of gas and air containing 4 per cent of gas (methane and ethane) and 20 pounds of bituminous coal dust, 18 pounds of which is to be placed on shelves along the sides of the first 20 feet of the gallery and 2 pounds to be so placed that it will be stirred up by an air current in such manner that all or part of it will be suspended in the first division of the gallery. An explosive is considered to have passed the test if no one of the five shots ignites this mixture.

DEFINITION OF PERMISSIBLE EXPLOSIVE.

An explosive is called a permissible explosive when it is similar in all respects to the sample that passed certain tests by the United States Bureau of Mines and when it is used in accordance with the conditions prescribed by this bureau.

[a] Two pounds of clay stemming are used with slow-burning explosives.

Subject to the conditions and provisions stated above, the following explosives are classified as permissible explosives:

TABLE 1.—*Permissible explosives tested prior to Mar. 1, 1913.*

Brand.	Class designation.	When used with detonators, preferably electric detonators, of not less efficiency than—	Manufacturer.
Ætna coal powder A	Class 4..	No. 6	Ætna Powder Co., Chicago, Ill.
Ætna coal powder AA	Class 1a.	do	Do.
Ætna coal powder B	Class 4..	do	Do.
Ætna coal powder C	do	do	Do.
Bental coal powder No. 1-A	Class 1a.	do	Independent Powder Co., Joplin, Mo.
Bental coal powder No. 2	do	No. 7	Do.
Bituminite No. 1	Class 4..	No. 6	Jefferson Powder Co., Birmingham, Ala.
Bituminite No. 3	do	do	Do.
Bituminite No. 4	do	do	Do.
Bituminite No. 5	Class 1a.	do	Do.
Black Diamond No. 2-A	Class 4..	do	Illinois Powder Manufacturing Co., St. Louis, Mo.
Black Diamond No. 3-A	do	do	Do.
Black Diamond No. 5	Class 1a.	do	Do.
Cameron mine powder No. 1-A	do	do	Cameron Powder Manufacturing Co., Emporium, Pa.
Cameron mine powder No. 2-A	do	do	Do.
Cameron mine powder No. 2-A, L. F	do	do	Do.
Cameron mine powder No. 3-A	Class 4..	do	Do.
Carbonite No. 1	do	do	E. I. du Pont de Nemours Powder Co., Wilmington, Del.
Carbonite No. 2	do	do	Do.
Carbonite No. 3	do	do	Do.
Carbonite No. 4	do	do	Do.
Carbonite No. 5	do	do	Do.
Carbonite No. 6	do	do	Do.
Coalite A	Class 1a.	do	Potts Powder Co., New York, N. Y.
Coalite X	do	do	Do.
Coalite No. 1	Class 4..	do	Do.
Coalite No. 2-D	do	do	Do.
Coalite No. 2-D. L	do	do	Do.
Coalite No. 2-M, L. F	do	do	Do.
Coalite No. 3-X	Class 1a.	do	Do.
Coalite No. 3-XA	do	do	Do.
Coalite No. 3-XB	do	do	Do.
Coalite No. 3-XC	do	do	Do.
Coal Special No. 1	Class 4..	do	Keystone National Powder Co., Emporium, Pa.
Coal Special No. 2	do	do	Do.
Coal Special No. 2-W	do	do	Do.
Coal Special No. 3-C	do	do	Do.
Collier powder B, N. F	Class 1a.	do	Do.
Collier powder KN	do	do	Do.
Collier powder No. X	do	do	Do.
Collier powder X, L. F	do	do	Do.
Collier powder No. 2	Class 4..	do	Do.
Collier powder No. 5	Class 1a.	do	Do.
Collier powder No. 5-L. F	do	do	Do.
Collier powder No. 5, special	do	do	Do.
Collier powder No. 6-L. F	Class 4..	do	Do.
Collier No. 9	Class 1a.	do	Do.
Collier powder No. 11	do	do	Do.
Cronite No. 1	do	do	G. R. McAbee Powder & Oil Co., Pittsburgh, Pa.
Cronite No. 5	do	No. 7	Do.
Detonite special	do	do	The King Powder Co., Cincinnati, Ohio.
Eureka No. 2	Class 2..	No. 6	G. R. McAbee Powder & Oil Co., Pittsburgh, Pa.
Fort Pitt mine powder No. 1	Class 4..	do	Fort Pitt Powder Co., Pittsburgh, Pa.
Fuel-ite No. 1	do	do	Burton Powder Co., Pittsburgh, Pa.
Fuel-ite No. 2	do	do	Do.
Fuel-ite No. 3	Class 1a.	do	Do.
Giant A low-flame dynamite	Class 2..	do	Giant Powder Co. (Consolidated), Giant, Cal.
Giant B low-flame dynamite	do	do	Do.
Giant C low-flame dynamite	do	do	Do.
Giant coal-mine powder No. 5	Class 1a.	do	Do.
Giant coal-mine powder No. 6	Class 2..	do	Do.
Giant coal-mine powder No. 7	do	do	Do.
Giant coal-mine powder No. 8	do	do	Do.

TABLE 1.—*Permissible explosives tested prior to Mar. 1, 1913*—Continued.

Brand.	Class designation.	When used with detonators, preferably electric detonators, of not less efficiency than—	Manufacturer.
Guardian A	Class 4..	No. 7.....	Independent Powder Co., Joplin, Mo.
Guardian No. 2	Class 1a.	No. 6.....	Do.
Guardian No. 3	...do....	.....do.....	Do.
Guardian coal powder B	Class 4..	No. 7.....	Do.
Hecla No. 2	Class 1a.	.....do.....	E. I. du Pont de Nemours Powder Co., Wilmington, Del.
Kanite A	Class 1b.	No. 8.....	W. H. Blumenstein Chemical Works, Pottsville, Pa.
Lomite No. 1	Class 2..	No. 6.....	G. R. McAbee Powder & Oil Co., Pittsburgh, Pa.
Meteor AXXO	...do....	.....do.....	E. I. du Pont de Nemours Powder Co., Wilmington, Del.
Mine-ite A	Class 4..	.....do.....	Burton Powder Co., Pittsburgh, Pa.
Mine-ite A-2	...do....	.....do.....	Do.
Mine-ite B	...do....	.....do.....	Do.
Mine-ite B-2	...do....	.....do.....	Do.
Mine-ite No. 5-D	Class 1a.	.....do.....	Do.
Monobel No. 1	...do....	.....do.....	E. I. du Pont de Nemours Powder Co., Wilmington, Del.
Monobel No. 2	...do....	.....do.....	Do.
Monobel No. 3	...do....	.....do.....	Do.
Monobel No. 4	...do....	.....do.....	Do.
Monobel No. 5	...do....	.....do.....	Do.
Monobel No. 6	...do....	.....do.....	Do.
Monobel No. 7	...do....	.....do.....	Do.
Nitro low-flame No. 1	Class 4..	.....do.....	Nitro Powder Co., Kingston, N. Y.
Nitro low-flame No. 2	...do....	.....do.....	Do.
Trojan coal powder H	Class 3..	No. 6.....	Pennsylvania Trojan Powder Co., Allentown, Pa.
Trojan coal powder I	...do....	.....do.....	Do.
Trojan coal powder J	...do....	.....do.....	Do.
Tunnelite B	Class 1a.	.....do.....	G. R. McAbee Powder & Oil Co., Pittsburgh, Pa.
Tunnelite C	...do....	.....do.....	Do.
Tunnelite No. 5	Class 4..	.....do.....	Do.
Tunnelite No. 6	...do....	.....do.....	Do.
Tunnelite No. 6-L. F	...do....	.....do.....	Do.
Tunnelite No. 7	...do....	.....do.....	Do.
Tunnelite No. 8	...do....	.....do.....	Do.
Tunnelite No. 8-L. F	...do....	.....do.....	Do.

The tests now prescribed as those a permissible explosive must have passed are those given on pages 302–3. But even the explosives that have passed those tests and are published as permissible explosives are to be considered as permissible explosives only when used under the following conditions:

1. That the explosive is in all respects similar to the sample submitted by the manufacturer for test.

2. That detonators—preferably electric detonators—are used of not less efficiency than those prescribed, namely, those consisting by weight of 90 parts of mercury fulminate and 10 parts of potassium chlorate (or their equivalents).

3. That the explosive, if frozen, shall be thoroughly thawed in a safe and suitable manner before use.

4. That the quantity used for a shot does not exceed 1½ pounds (680 grams), properly tamped with clay or other noncombustible stemming.

It must not be supposed that an explosive that has once passed the above-mentioned tests and has been published in lists of permissible explosives is thereafter to be considered a permissible explosive, regardless of its condition or the way in which it is used. Thus, for example, an explosive named in the permissible list, if kept in a moist place until it undergoes a change in character is no longer to be considered a permissible explosive. If used in a frozen or partly frozen condition, it is not when so used a permissible explosive. If used in excess of the quantity specified

(1½ pounds), it is not when so used a permissible explosive. And when the other conditions have been met, it is not a permissible explosive if fired with a detonator of less efficiency than that prescribed.

Moreover, even when all the prescribed conditions have been met, no permissible explosive should necessarily be considered as *permanently* being a permissible explosive, but any permissible explosive when used under the prescribed conditions may properly continue to be considered a permissible explosive until notice of its withdrawal or removal from the list has been officially published, or until its name is omitted from a later list published by the Bureau of Mines.

Furthermore, the manufacturers of a permissible explosive may withdraw it at any time when introducing a new explosive of superior qualities. And after further experiments and conferences the Bureau of Mines may find it advisable to adopt additional and more severe tests to which all permissible explosives may be subjected, in the hope that through the use of such explosives only as may pass the more severe tests the lives of miners may be better safeguarded.

CLASSIFICATION OF PERMISSIBLE EXPLOSIVES.

In Table 2 the permissible explosives are arranged in four classes, the rate of detonation of each explosive being given.

TABLE 2.—*Classes and rate of detonation of permissible explosives.*

Class and brand of explosive.	Rate of detonation in 1½ by 8 inch cartridge.	
	Feet per second.	Meters per second.
CLASS 1.—AMMONIUM NITRATE.		
Subclass a.		
Ætna coal powder AA	7,880	2,403
Bental coal powder No. 1-A	7,690	2,343
Bental coal powder No. 2	4,750	1,447
Bituminite No. 5	9,120	2,782
Black Diamond No. 5	6,020	1,835
Cameron mine powder No. 1-A	10,950	3,339
Cameron mine powder No. 2-A	10,520	[a] 3,208
Cameron mine powder No. 2-A, L. F	10,950	3,338
Coalite A	8,870	2,705
Coalite X	9,000	2,745
Coalite No. 3-X	7,290	2,223
Coalite No. 3-XA	10,420	3,176
Coalite No. 3-XB	11,290	3,441
Coalite No. 3-XC	8,320	2,536
Collier powder B, N. F	10,950	3,338
Collier powder KN	11,610	3,539
Collier powder No. X	7,820	2,384
Collier powder X, L. F	9,990	3,047
Collier powder No. 5	8,280	2,524
Collier powder No. 5-L. F	9,370	2,858
Collier powder No. 5, special	8,330	2,541
Collier No. 9	6,690	2,040
Collier powder No. 11	11,680	3,561
Cronite No. 1	10,490	3,198
Cronite No. 5	8,090	2,466
Detonite special	10,840	[b] 3,305
Fuel-ite No. 3	8,590	2,620
Giant coal-mine powder No. 5	10,230	3,118
Guardian No. 2	8,980	2,738
Guardian No. 3	7,660	2,336
Hecla No. 2	13,990	4,264
Mine-ite No. 5-D	8,720	2,658
Monobel No. 1	11,700	3,568
Monobel No. 2	9,870	3,009
Monobel No. 3	7,260	2,212
Monobel No. 4	8,980	2,738
Monobel No. 5	6,760	2,061
Monobel No. 6	10,380	3,165
Monobel No. 7	10,050	3,065
Tunnelite B	9,990	3,047
Tunnelite C	10,060	3,068

[a] 1½-inch cartridges used. [b] 1¼-inch cartridges used.

TABLE 2.—*Classes and rate of detonation of permissible explosives*—Continued.

Class and brand of explosive.	Rate of detonation in 1¼ by 8 inch cartridge.	
	Feet per second.	Meters per second.
CLASS 1.—AMMONIUM NITRATE—Continued.		
Subclass b.		
Kanite A	10,730	[a]3,270
CLASS 2.—HYDRATED.		
Eureka No. 2	10,720	3,269
Giant A low-flame dynamite	8,140	2,481
Giant B low-flame dynamite	9,100	2,773
Giant C low-flame dynamite	8,860	2,702
Giant coal-mine powder No. 6	11,540	3,519
Giant coal-mine powder No. 7	10,830	3,301
Giant coal-mine powder No. 8	10,830	3,301
Lomite No. 1	13,940	4,250
Meteor AXXO	8,410	2,563
CLASS 3.—ORGANIC NITRATE (OTHER THAN NITROGLYCERIN).		
Trojan coal powder H	9,420	2,872
Trojan coal powder I	9,620	2,934
Trojan coal powder J	10,590	3,230
CLASS 4.—NITROGLYCERIN.		
Ætna coal powder A	12,620	3,848
Ætna coal powder B	9,870	3,008
Ætna coal powder C	7,010	2,138
Bituminite No. 1	12,800	3,901
Bituminite No. 3	9,330	2,843
Bituminite No. 4	7,540	2,298
Black Diamond No. 2-A	12,600	3,842
Black Diamond No. 3-A	11,160	3,402
Cameron mine powder No. 3-A	10,930	3,333
Carbonite No. 1	10,950	3,338
Carbonite No. 2	11,380	3,470
Carbonite No. 3	8,710	2,656
Carbonite No. 4	7,670	2,339
Carbonite No. 5	10,140	3,092
Carbonite No. 6	7,490	2,285
Coalite No. 1	7,930	2,418
Coalite No. 2-D	8,430	2,571
Coalite No. 2-D. L	7,340	2,237
Coalite No. 2-M, L. F	9,220	2,811
Coal special No. 1	11,800	3,598
Coal special No. 2	10,240	3,123
Coal special No. 2-W	11,590	3,534
Coal special No. 3-C	9,760	2,977
Collier powder No. 2	8,870	2,704
Collier powder No. 6-L. F	9,660	2,944
Fort Pitt mine powder No. 1	11,230	3,424
Fuel-ite No. 1	11,150	3,400
Fuel-ite No. 2	8,240	2,512
Guardian A	10,130	3,089
Guardian coal powder B	8,960	2,733
Mine-ite A	11,290	3,443
Mine-ite A-2	10,670	3,252
Mine-ite B	9,540	2,908
Mine-ite B-2	7,410	2,260
Nitro low-flame No. 1	13,380	4,079
Nitro low-flame No. 2	14,560	4,439
Tunnelite No. 5	8,400	2,562
Tunnelite No. 6	9,670	2,947
Tunnelite No. 6-L. F	9,620	2,933
Tunnelite No. 7	8,720	2,659
Tunnelite No. 8	8,020	2,446
Tunnelite No. 8-L. F	8,720	2,658

[a] 1¼-inch cartridges used.

PUBLICATIONS ON MINE ACCIDENTS AND TESTS OF EXPLOSIVES.

The following Bureau of Mines publications may be obtained free by applying to the Director, Bureau of Mines, Washington, D. C.

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 15. Investigations of explosives used in coal mines, by Clarence Hall, W. O. Snelling, and S. P. Howell, with a chapter on the natural gas used at Pittsburgh, by G. A. Burrell, and an introduction by C. E. Munroe. 1911. 197 pp., 7 pls., 5 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 42. The sampling and examination of mine gases and natural gas, by G. A. Burrell, 1912. 106 pp., 2 pls., 21 figs.

BULLETIN 44. First national mine-safety demonstration, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 46. An investigation of explosion-proof motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1912. 50 pp., 3 pls., 7 figs.

BULLETIN 51. The analysis of black powder and dynamite, by W. O. Snelling and C. G. Storm. 1913. 80 pp., 5 pls., 5 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 56. First series of coal-dust tests in the experimental mine, by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 59. Investigations of detonators and electric detonators, by Clarence Hall and S. P. Howell. 1913. 79 pp., 10 pls., 5 figs.

BULLETIN 62. Record of the national mine-rescue and first-aid conference, Pittsburgh, Pa., September 23–26, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 65. Oil and gas wells through workable coal beds; papers and discussions, by G. S. Rice, O. P. Hood, and others. 1913. 101 pp., 1 pl., 11 figs.

TECHNICAL PAPER 4. The electrical section of the Bureau of Mines, its purpose and equipment, by H. H. Clark. 1911. 12 pp.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigations of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 12. The behavior of nitroglycerin when heated, by W. O. Snelling and C. G. Storm. 1912. 14 pp., 1 pl., 2 figs.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp. 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 23. Ignition of gas by miniature electric lamps, by H. H. Clark. 1912. 5 pp.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp. 10 figs.

TECHNICAL PAPER 52. Permissible explosives tested prior to March 1, 1913, by Clarence Hall. 1913. 11 pp.

MINERS' CIRCULAR 3. Coal-dust explosions, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp., 5 figs.

MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 9. Accidents from falls of roof and coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.

MINERS' CIRCULAR 12. The use and care of miners' safety lamps, by J. W. Paul. 1913. 16 pp., 4 figs.

MINERS' CIRCULAR 15. Rules for mine-rescue and first-aid field contests, by J. W. Paul. 1913. 12 pp.

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Bulletin 67

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

ELECTRIC FURNACES
FOR
MAKING IRON AND STEEL

BY

DORSEY A. LYON
AND
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ELECTRIC FURNACES FOR MAKING IRON AND STEEL.

By Dorsey A. Lyon and Robert M. Keeney.

THE ELECTRIC FURNACE IN PIG-IRON MANUFACTURE.

By David A. Lyon.

INTRODUCTION.

In the inquiries and investigations that the Bureau of Mines is making with a view to increasing safety, efficiency, and economic development in the metallurgical industries, the application of electricity to various processes, and especially to those in the manufacture of iron and steel, is being given attention. Some results of the work already done are presented in this bulletin, which gives a historical review of the development of electric furnaces for making iron and steel, and discusses the problems that remain to be solved in the use of electric furnaces for the smelting of iron ores and the production of pig iron at a profit on a commercial scale.

DEVELOPMENT OF THE ELECTRIC FURNACE FOR SMELTING IRON.

In 1898 Capt. Stassano, of Italy, patented an electric furnace for smelting iron ores.[a] In 1900 the production of ferro-alloys in the electric furnace was begun, and at the present time practically all ferro-alloys are made in electrically heated furnaces.

INVESTIGATIONS OF THE CANADIAN COMMISSION.

INVESTIGATION OF 1904.

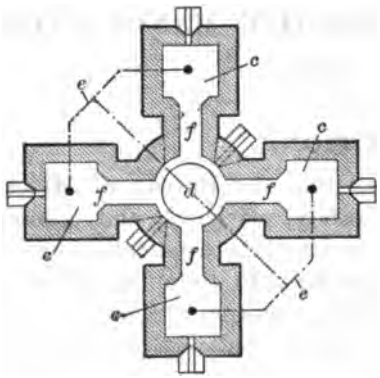
The work of Stassano and the successful production of ferro-alloys in the electric furnace were brought to the attention of the Canadian Government by Eugene Haanel, Government superintendent of mines. As is well known, Canada is not so favored with abundant high-grade ores and good coking coal as is the United States, and so it seemed to Haanel and his associates that the electric furnace might lead to the solution of their problem in Canada, namely, the treatment of low-grade ores, and of those ores high in phosphorus and sulphur.

Ultimately, through the efforts of Haanel, a commission was appointed "to proceed to Europe for the purpose of investigating

[a] Stansfield, A., The electric furnace, 1907, p. 13.

and reporting upon the different electrothermic processes employed in the smelting of iron ores and the making of the different classes of steel, then in operation or in process of development, in Italy, France, and Sweden."

This commission left Ottawa for England on January 21, 1904. When the commission arrived in England, F. W. Harbord, the well-known metallurgist, was engaged to act as metallurgist to it. Although the commission visited several plants in Europe, yet in none of them was direct smelting being attempted, nor were most of them in any way adapted for making experimental tests of direct smelting. While the commission was at La Praz, France, Dr. Héroult was kind enough to demonstrate that it was possible to produce iron from the ore in an electric furnace.



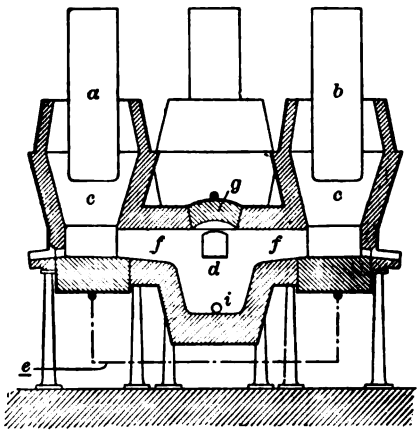


FIGURE 1.—Sectional plan and elevation of Keller furnace. Used by the Canadian commission in making tests at Livet, France.

EXPERIMENTS WITH KELLER FURNACE, LIVET, FRANCE.

At Livet, France, however, the commission found at the works of Keller, Leleux and Co. an electrical installation well suited for smelting tests of iron ores. Figure 1 shows the type of furnace used for making these tests. It consisted of two or more furnaces with vertical electrodes, connected by a central well, the current flowing from each furnace through the vertical electrodes *a* and *b*. In the tests made by the commission the iron ore, fluxes, and coke were broken to such a size that all would pass through a 1½-inch ring. They were then mixed and "charged into the furnace in the annular space between the electrode and the walls of the furnace." In this connection it should be noted that the report of the commission clearly states that in these experiments the reduction was accomplished by means of solid carbon, as is shown by the following statement: "The heat, generated rapidly, raised the temperature and en-

abled the carbon mixed with the ore to reduce it to the metallic state, and as the temperature rose the metal fused and collected on the sole of the furnace." Later reference to this point is made. The gases resulting from the operation were allowed to escape and burn at the top of the furnace.

At Livet the commission made two separate experiments, the first lasting 55 hours and the second 48 hours. In conducting these experiments the materials were weighed and the weights checked. Especial attention was also given to such other points as were necessary to obtain accurate data in regard to the following matter:

The output of pig iron for a given consumption of electric energy.
The yield of metal per ton of ore charged.
The quantity of coke required as a reducing agent.
The quality of pig iron obtained, with especial reference to its suitability for (*a*) steel manufacture by (1) Bessemer or Siemen's acid process or (2) Bessemer or Siemen's basic process; and (*b*) foundry purposes.

The tests, performed on March 19, 20, and 21, 1904, marked the real beginning of the present practice of the reduction of iron ores in an electric furnace, for although, as will be seen later, the furnaces in use for this purpose at present differ considerably in construction from the Keller furnace, the principle employed is the same—namely, the ore, flux, and reducing agent are fed around the electrodes, and the heat generated by the current passing through the charge raises the temperature of the ore and fluxes to their melting point, and thus enables the carbon to reduce the ore with which it is mixed.

PROBLEMS NOT SOLVED BY COMMISSION OF 1904.

After the report[a] of the commission had been carefully studied it was evident that further data would have to be obtained in order to establish with some degree of exactitude the quantity of electric energy required per ton of product and the consumption of electrode. Also, the following important questions referring to Canadian conditions were either not taken up or were left in doubt by the Livet experiments:

Could magnetite, which is Canada's chief ore and is to some extent a conductor of electricity, be successfully and economically smelted by the electrothermic process?
Could iron ores with comparatively high sulphur content, but not containing manganese, be made into pig iron of marketable composition?
The experiments made at Livet with charcoal as a reducing agent in substitution for coke having failed, could the process be so modified that charcoal could be substituted for coke?

[a] Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe: Mines Branch, Department of Interior, Canada, 1904.

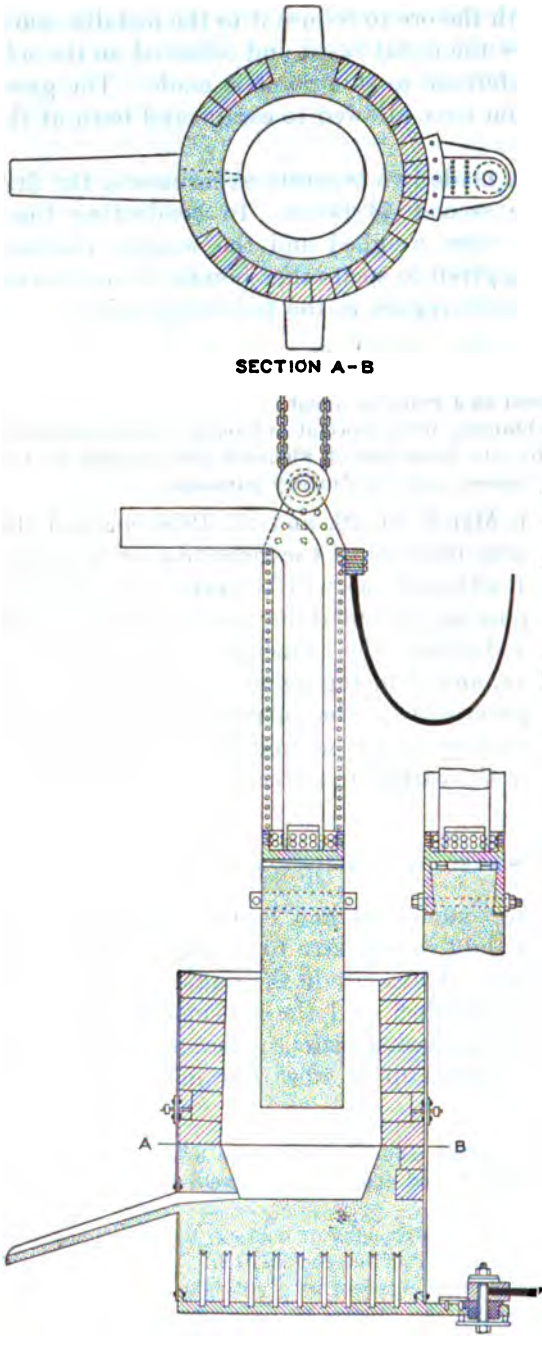


FIGURE 2.—Experimental furnace used at Sault Ste. Marie, Ontario.

The last problem was especially important, as charcoal and possibly peat coke would constitute Canada's home products, whereas coal coke for metallurgical processes has to be imported into several of the Provinces.

EXPERIMENTS AT SAULT STE. MARIE, ONTARIO, 1906.

A sum of $15,000 was placed at the disposal of the Canadian commission for prosecuting experiments at Sault Ste. Marie, Ontario, in 1906. Owing to the advantages offered, the experimental furnace[a] was erected at the plant of the Lake Superior Corporation at Sault Ste. Marie, Ontario, under the direction of Erik Nystrom, a member of the staff of the mines branch of the interior department, who was later prominently connected with the work at Trollhättan, Sweden. The furnace is shown in section in figure 2.

The experiments at Sault Ste. Marie were continued for several weeks. The report made to the Canadian Government by the superin-

[a] Haanel, Eugene, Report on the experiments made at Sault Ste. Marie, Ontario, under Government auspices, on smelting of Canadian iron ores by the electrothermic process: Mines Branch, Department of Interior, Canada, 1907, p. 2.

tendent of mines contained substantially the following conclusions relative to the experiments:

Canadian magnetite ores can be as economically smelted as hematites by the electrothermic process.

Ores of high sulphur content can be made into pig iron containing only a few thousandths of 1 per cent of sulphur.

The silicon content can be varied as required for the class of pig that is to be produced.

Charcoal, which can be cheaply produced from mill refuse or wood that could not otherwise be utilized, and peat coke can be substituted for coke without being briquetted with the ore.

A ferronickel pig practically free from sulphur and of fine quality can be produced from roasted nickeliferous pyrrhotite.

Titaniferous iron ores containing up to 5 per cent of titanium can be successfully treated by the electrothermic process. This conclusion is based upon an experiment made with an ore containing 17.82 per cent of titanic acid, yielding a pig iron of good quality.

The electrical horsepower-year used per ton of pig produced during these experiments was about 0.277. As noted later, an average of 18 months' running at Trollhättan, Sweden, indicated that the electrical horsepower-year used per ton of pig produced was 0.340.

After the Government had discontinued its tests the experimental plant was acquired by the Lake Superior Corporation, and for a time was employed for the semicommercial production of ferronickel pig, but up to the present time no other electric pig-iron reduction furnace plants have been installed in Canada. This lack of plants is not because the reduction of iron ores in the electric furnace has not met the expectations of the commission, but because the peculiar economic and geographic conditions in Canada have not seemed to warrant, to the present time, the introduction of the electric furnace for the production of pig iron.

DEVELOPMENT OF THE ELECTRIC IRON-REDUCTION FURNACE IN SWEDEN.

The development of the electric iron-reduction furnace in Sweden has been due to the following reasons:

In Sweden the conditions are somewhat analogous to those found in Canada, that is, there are iron ores, but no coal for coking. There is, however, this difference between the conditions in Canada and in Sweden—in Sweden, as also in California, the ores are for the most part high in their iron content and rather free from impurities. Moreover, as has been indicated by Sundbarg[a], the iron industry has been well established in Sweden for some hundreds of years. The ores have been smelted in a blast furnace, charcoal for the most part having been used as a reducing agent, and Swedish charcoal iron is known the world over for its purity. However, it has long been

[a] Sundbarg, A. G., Sweden, its people and its industries, 1904.

apparent to those familiar with the situation that there would have to be some innovation in order to enable Swedish iron manufacturers to keep the cost of production down. Each year has seen an increase in the cost of charcoal, because wood for making charcoal is becoming scarce owing to the depletion of the forests and the increased production of wood pulp. These facts caused Swedish engineers to turn their attention to the possibilities of electric smelting.

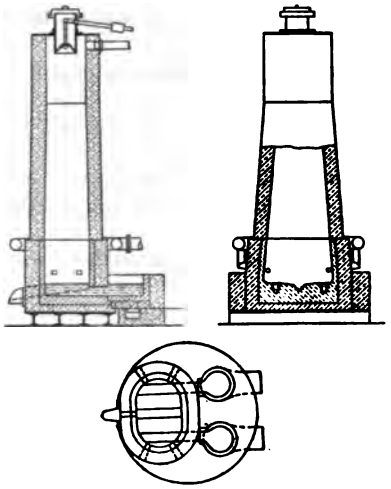


FIGURE 3.—First type of shaft furnace used in experiments of Grönwall, Lindblad, and Stalhane.

EXPERIMENTS OF GRÖNWALL, LINDBLAD, AND STALHANE, DOMNARFVET, SWEDEN.

The first electric-furnace smelting experiments made in Sweden were conducted by Grönwall, Lindblad, and Stalhane. The types of furnaces used by them are shown in figures 3, 4, and 5. The construction of the furnace shown in figure 3, is described by Yngstrom[a] as follows:

> It was a shaft furnace with the hearth lined with stamped silica. In the bottom of the hearth there were three channels: One in the middle leading to the tap hole for the pig iron, and one on each side. The latter communicated with two receptacles placed outside the shaft and filled with iron. The bottom of these receptacles was formed by blocks of granite packed on copper plates and connected with the electric-current supply. The furnace was started with an air blast as an ordinary blast furnace, and when sufficient iron had collected on the hearth the air blast was cut off and the electric current turned on. The

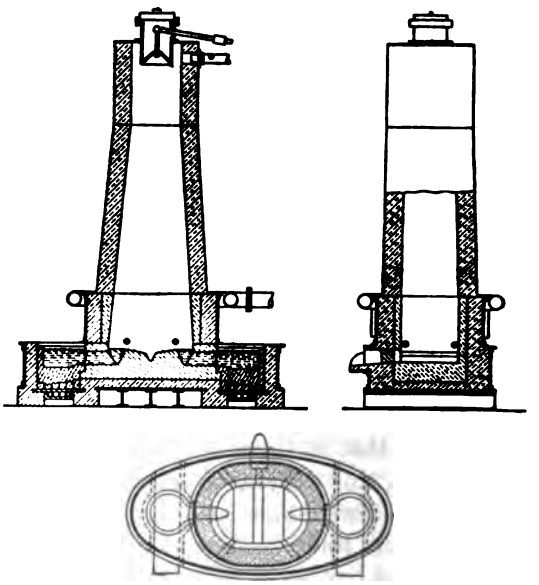


FIGURE 4.—Second type of shaft furnace used by Grönwall, Lindblad, and Stalhane.

[a] Yngstrom, Lars, Electric production of iron from iron ore at Domnarfvet, Sweden: Engineer (London), vol. 109, Feb. 25, 1910, p. 206.

idea was that the current entering the furnace through the iron by one of the channels would melt the charge in its passage to the opposite channel. It was possible to work the furnace in this manner during short periods, but it proved impossible to make the hearth durable. In part this was due to the fact that at the high temperatures used the silica became conductive of electricity. The furnace was therefore reconstructed in the manner shown in figure 4. The general principle of the design was the same as in the first furnace, but the lower portion of the furnace was of more solid construction. The electric current was introduced at opposite sides of the furnace. The most important alteration was, however, that the hearth was lined with magnesite brick. In consequence the hearth lasted somewhat better than before, but, in general, the same difficulties remained, and even the magnesite proved to be a fairly good conductor of electricity at high temperatures.

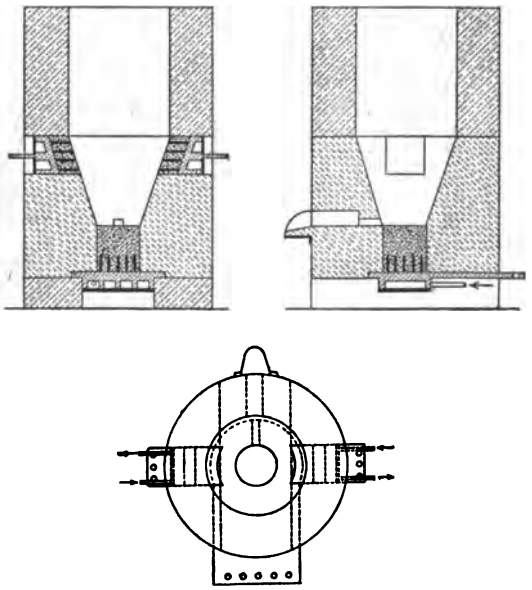


FIGURE 5.—Third type of furnace used by Grönwall, Lindblad, and Stalhane.

Finding that the types of furnaces above described were not suited to the work of reduction, Grönwall, Lindblad, and Stalhane turned their attention to the development of the type that later proved successful and is now being successfully operated in Sweden and Norway. In the work of developing this furnace to a feasible and commercial success these men were ably assisted by the engineers and capitalists of Sweden. An agreement was made with the Trafikaktiebolaget Grangesberg Oxelosund which enabled them to carry out their experiments on a large scale at the Domnarfvet Iron Works. Although the preliminary work on the furnace at Domnarfvet was begun in April, 1906, the furnace was not operated until April, 1907. Experimental work was conducted with it during the summer of 1907. The furnace is shown in figure 6. As the shaft of the furnace is low and open at the top, large quantities of charcoal were consumed at the top of the open shaft, and the gas escaping from the furnace consisted almost entirely of carbon monoxide. Based on the data

obtained during the operation of this furnace, a new furnace was constructed. This furnace, which is shown in figure 7, was higher, was closed at the top, and was so constructed as to permit the return to the crucible of a part of the gas passing from the top of the shaft. The furnace has been described in detail by Yngstrom in his report.[a]

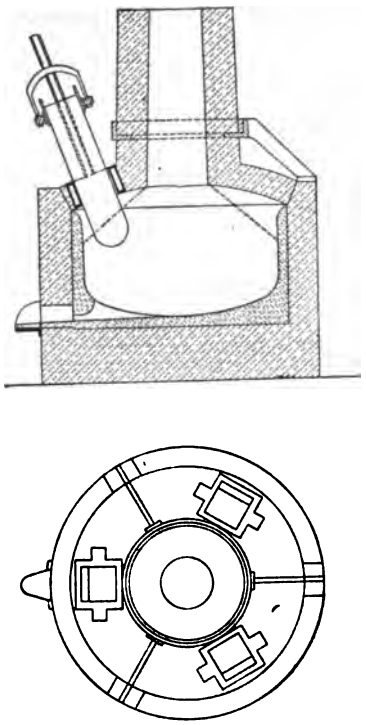


FIGURE 6.—First type of furnace used at Domnarfvet, Sweden.

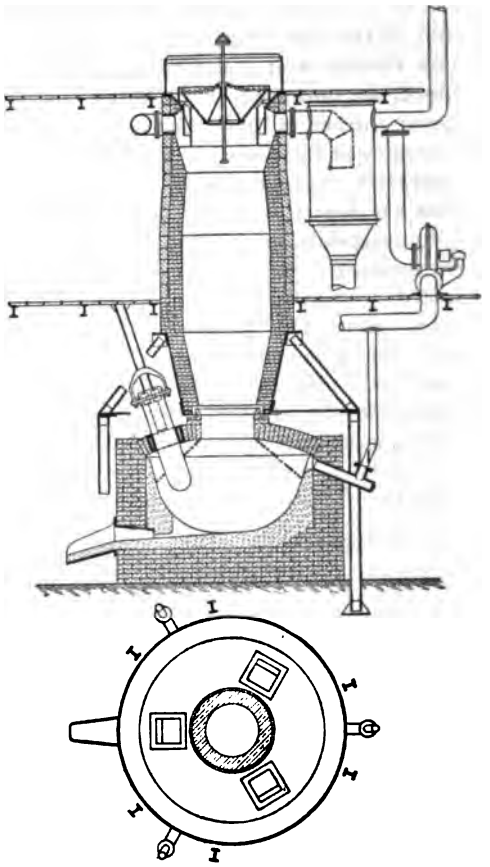


FIGURE 7.—Second type of furnace used at Domnarfvet, Sweden.

EXPERIMENTS OF THE JERN-KONTORET, TROLLHÄTTAN, SWEDEN.

The experimental runs conducted at Domnarfvet satisfied those interested in the undertaking so well that it was decided to go a step farther and to construct and perfect a furnace of a size suitable for commercial purposes.

[a] Yngstrom, Lars, Electric production of iron from iron ore at Domnarfvet, Sweden: Engineer (London), vol. 109, Mar. 4, 1910, p. 237.

DESCRIPTION OF FURNACE.

This work was undertaken by the Jern-Kontoret, an association of the ironmasters of Sweden. Realizing the importance of the work to the iron industry of that country, the association voted $90,000 for the purpose of putting up a plant and developing the process. The Swedish Government also assisted the project to the extent of furnishing power at a nominal figure from the plant at Trollhättan. Plans and sectional elevations of this furnace are shown in figures 8 to 11. The following description is taken from the official report delivered by the engineers to the Jern-Kontoret:[a]

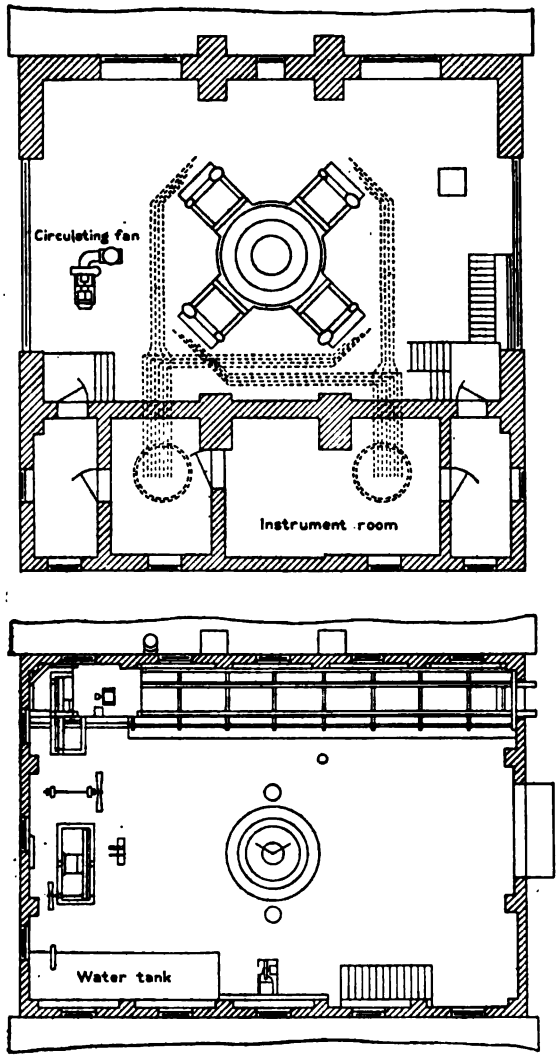


FIGURE 8.—Plan of furnace house at Trollhättan, Sweden.

The hearth is lined with "Hoganas" fire brick to a thickness of 360 mm. in the bosh and 450 mm. in the stack.

At the top of the shaft is a Tholander charging bell and cone, which is raised or lowered by means of a capstan driven by a motor.

The crucible rests on a concrete foundation. Like the shaft, it is surrounded by a sheet-iron shell § of an inch thick. At the top this shell is reinforced by an iron band to take up the pressure of the arched roof.

The four electrodes project through the roof in a slanting position at an angle of 65° to the horizontal. At the openings in the roof the electrodes are surrounded by cooling jackets of copper provided at the top with asbestos packing

[a] Leffler, J. A., and Odelberg, E., Redogörelse för Jern-Kontoret's Försöksverk I Trollhättan, May 31, 1911. Abstract: Iron and Coal Trades Rev., vol. 82, 1911, pp. 957, 1010.

to prevent the leakage of gas. The contacts for conducting the current to the electrode are arranged at the upper end and wedged between the electrodes and a holder of cast steel. This holder, in its turn, is supported on a frame movable up and down between two guides placed on each side of the electrode.

In this process provision is made for a circulation of the gases in such manner that gas is drawn by means of a fan from the gas outlets and blown into

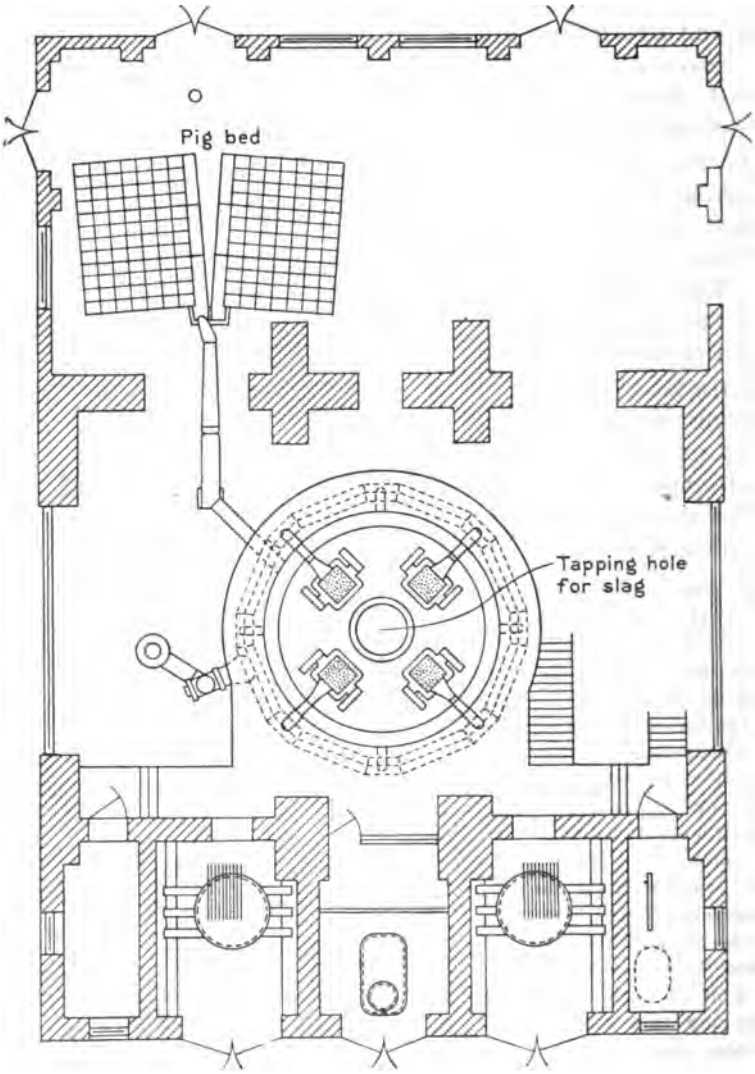


FIGURE 9.—Plan of furnace at Trollhättan.

the crucible. Consequently, the quantity of gas passing any given section of the shaft can be varied within certain limits by varying the speed of the fan.

This circulation of gas has two special objects, namely, (*a*) The gas blown into the crucible is there heated and gives off its heat to the charge in the shaft in passing up through it. Through this heating of the charge in the shaft a reduction of the ore by CO is facilitated, so that this gas is utilized to some

extent for the process. (*b*) The second purpose of the circulation of the gas is connected with the construction of the crucible, as the gas cools the roof and thus protects it from overheating. This cooling action of the gas is due in part to the absorption of heat or raising the temperature of the gas and in part to the decomposition of CO_2 and H_2O in the gas in contact with the incandescent carbon in the crucible.

A diagram of the electrical connections is shown in figure 12.

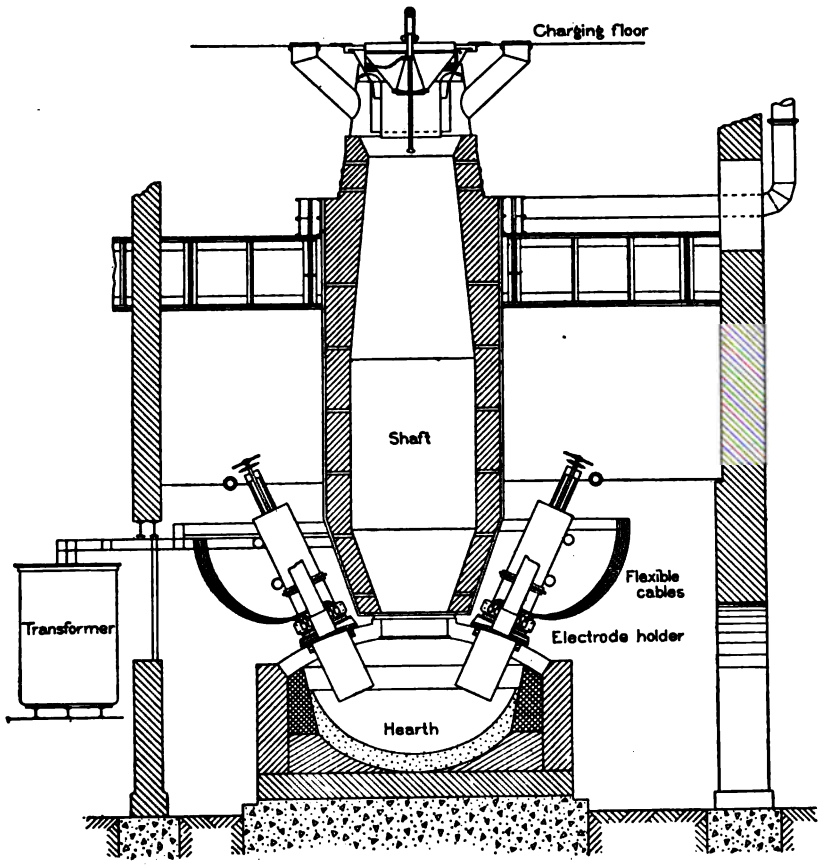


FIGURE 10.—Sectional elevation of furnace at Trollhättan.

CHANGES AFTER THE FIRST EXPERIMENTS.

Operations with this furnace were started November 15, 1910, and were continued without interruption until May 29, 1911. During this period various kinds of ore were smelted, and such grades of acid and basic slags were produced as seemed best suited to the treatment of each particular ore. The furnace worked well from the start, and was closed down that such alterations might be made as experience had demonstrated would be beneficial, and also that the crucible

might be relined and a new roof built, as the frequent alternation of acid and basic slags had damaged the brickwork of the crucible. The

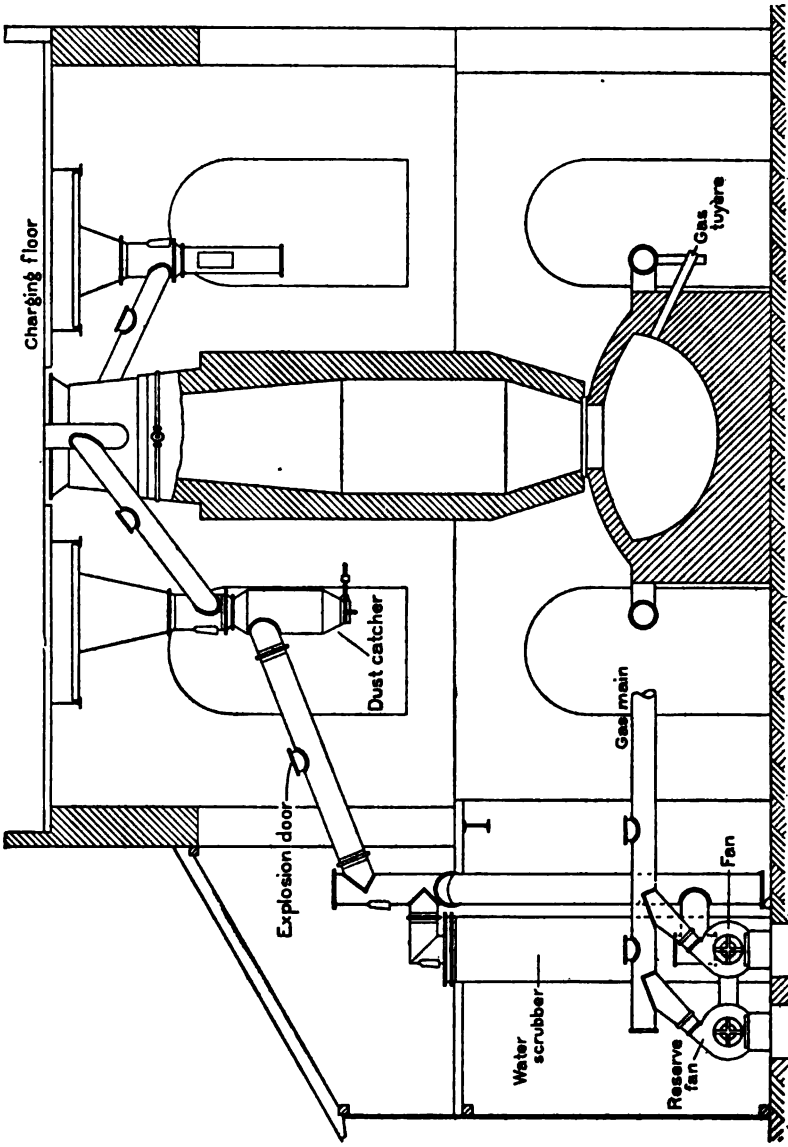


FIGURE 11.—Sectional elevation of furnace at Trollhättan, showing arrangement of gas circulation.

principal changes and additions that were made after the closing down of the furnace were as follows:

A system was installed for washing the gas which is returned to the crucible, so as to free the gas from dust particles before it reached the fan that caused the circulation.

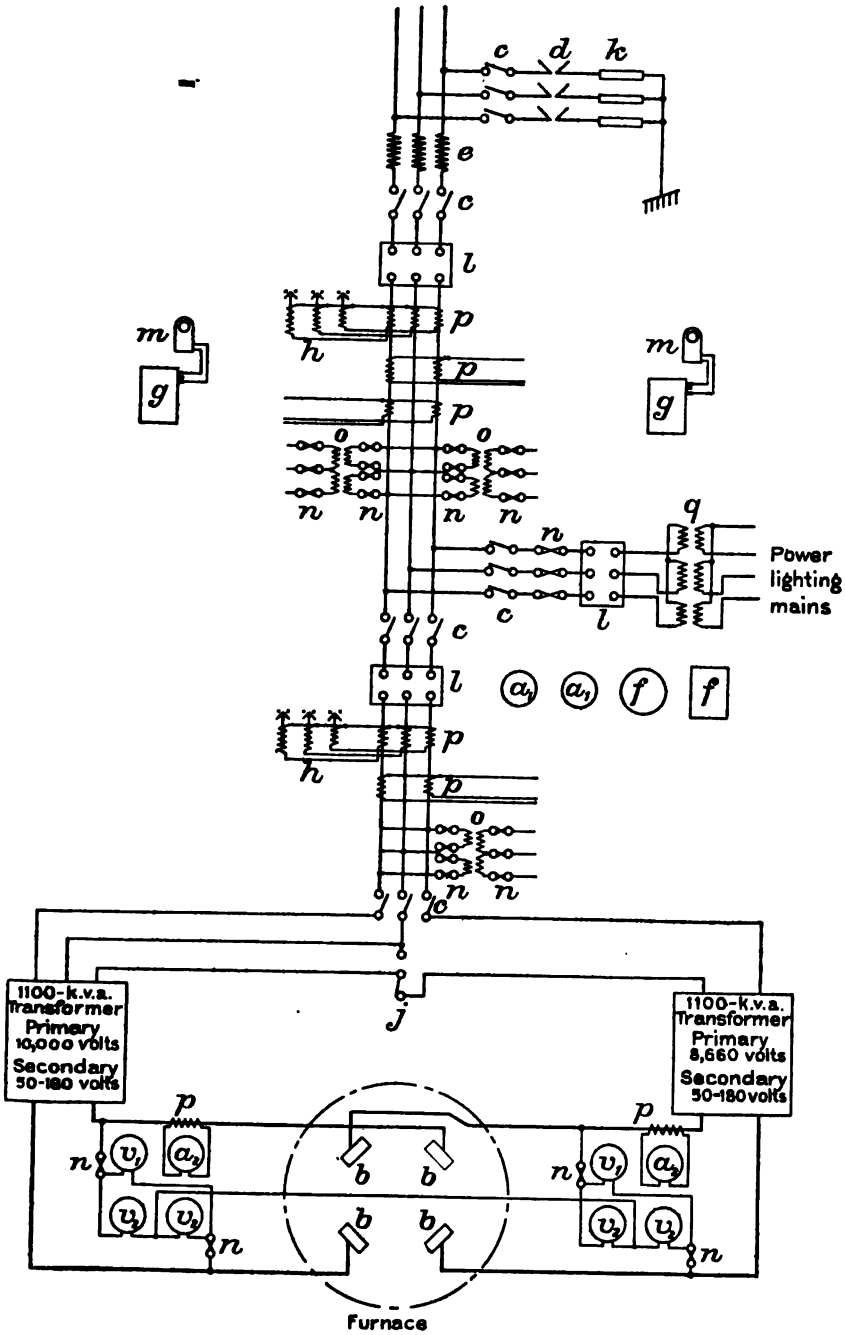


FIGURE 12.—Diagram of electrical connections of furnace at Trollhättan. *a*, ammeters; *b*, electrodes; *c*, main switches; *d*, lightning arresters; *e*, induction coil; *f*, kilowatt meters; *g*, kilowatt-hour meters; *h*, maximum relay; *j*, switch; *k*, oil resistance; *l*, automatic switches; *m*, clock control; *n*, fuses; *o*, *p*, *q*, transformers; *v*, voltmeters.

Electrode contact jackets were constructed and installed which permitted the electric current to be introduced to the electrodes at the point where they pass through the roof of the crucible rather than at the end of the electrode as was formerly done (fig. 10).

Round electrodes, 23§ inches in diameter, were substituted for square electrodes. The latter were built of four carbons, of about 12 by 12 inch section.

The electrodes were threaded so as to permit their being joined by means of screw nipples, thus avoiding the waste of butt ends, as formerly, as soon as the electrodes became too short for further use.

SUMMARY OF WORKING RESULTS AND ANALYSES.

The following tables present in condensed form the results obtained by the operation of the furnace from November 15, 1910, to April 9, 1911:

TABLE 1.—*Results of electric iron smelting at Trollhättan, Sweden.*

Item.	Nov. 15, 1910.[a]	Nov. 16, 1910, to Feb. 11, 1911.	Feb. 11 to Feb. 19, 1911.	Feb. 19 to Mar. 19, 1911.	Mar. 19 to Apr. 9, 1911.	Total.
Iron in ore, per cent	64.92	65.57	65.06	49.50	57.92	61.54
Iron in charge, per cent	59.80	62.10	62.56	42.42	53.06	57.00
Slag per ton of iron, kg	390.00	205.00	224.00	780.00	458.00	327.00
Material charged per hectoliter of charcoal, kg		66.49	71.13	90.31	69.88	70.77
Charcoal per ton of iron, hectoliters		24.22	22.47	26.10	26.97	24.79
Charcoal contents:						
Water, kg		69.1	50.8	59.8	40.2	60.9
Gas, kg		41.7	36.9	49.3	43.1	42.9
Ash, kg		11.8	11.0	13.2	17.2	12.8
Coke, kg		293.1	277.6	323.4	325.7	301.4
Total, kg		415.7	376.3	445.7	426.2	418.0
Time consumed in working, hours, minutes	7 50	2,009 56	184 32	639 18	506 34	3,348 10
Time consumed in interruptions, hours, minutes		105 39	4 58	20 57	22 11	153 45
Total time, hours, minutes	7 50	2,115 35	189 30	660 15	528 45	3,501 55
Average load, kilowatts	1,121	1,319	1,694	1,017	1,733	1,344
Total kilowatt-hours used	8,780	2,651,029	312,601	650,480	877,706	4,500,596
Kilowatt-hours per ton of iron	3,800	2,296	2,149	2,623	2,643	2,391
Iron per kilowatt-year, tons	2.31	3.82	4.08	3.34	3.31	3.66
Gross electrode consumption, kg		13,012	1,578	2,281	2,474	19,345
Net electrode consumption, kg		6,743	763	1,121	1,285	9,912
Gross electrode consumption per ton of iron, kg		11.24	10.84	9.19	7.45	10.28
Net electrode consumption per ton of iron, kg		5.83	5.24	4.52	3.87	5.27

[a] Furnace being filled.

[b] Term "net" represents 51.24 per cent of gross.

TABLE 2.—*Analyses*[a] *of ore and limestone used in electric iron smelting plant at Trollhättan, Sweden*

[From Iron and Coal Trades Review, June 9, 1911.]

Number of sample.[b]	Oxides.											Loss on ignition.	Total.	Elements.				
	Fe_2O_3	Fe_3O_4	FeO	MnO	MgO	CaO	Al_2O_3	TiO_2	SiO_2	P_2O_5	S			Fe	Si	Mn	S	P
	P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
1	0.24	.....	.....	0.35	0.31	54.32	Trace.	.....	1.68	Trace.	0.001	42.94	99.811	0.17	0.79	0.27	0.001	Trace.
2	1.41	91.05	.....	.16	1.12	.28	0.16	0.10	3.42	0.050	.011	.16	97.921	66.94	1.61	.12	.011	0.022
3	.....	64.80	2.95	7.30	3.17	3.95	1.16	.....	8.45	.016	.019	[c] 7.38	99.695	49.18	3.97	5.66	.019	.007
4	53.36	25.07	.....	.40	.60	.71	.59	.....	20.13	.034	.013	.27	101.177	55.45	9.47	.31	.013	.015
5	89.85	.....	2.49	.18	.85	.99	.42	.....	5.08	.914	.007	.59	100.471	64.80	2.39	.14	.007	.006
6	5.52	37.45	.....	.14	1.84	.53	.06	.48	3.47	.037	.012	.14	99.679	67.10	1.63	.11	.012	.016
7	80.88	.....	4.01	.63	2.57	.65	1.82	.....	7.31	.032	.017	1.54	99.459	59.68	3.44	.49	.017	.014
8	.....	79.25	1.42	.45	4.23	2.31	2.55	.....	7.53	.028	.023	1.31	99.501	58.51	3.68	.35	.023	.012
9	.....	73.31	.....	.22	4.73	4.29	2.67	.....	13.25	.025	.014	1.08	99.539	53.06	6.23	.17	.014	.011
10	.....	73.10	.....	.19	6.90	4.37	1.87	.....	11.19	.025	.011	2.17	99.826	52.90	5.26	.15	.011	.011
11	.....	71.69	.....	.12	7.35	1.34	1.27	.....	16.28	.025	.009	1.22	99.304	51.88	7.65	.09	.009	.011
12	2.92	78.02	.....	.14	.13	.82	1.34	.....	15.05	.027	.009	.90	99.356	58.50	7.06	.11	.009	.012
13	.....	70.13	.....	.12	7.90	7.05	.90	.....	10.63	.029	.011	5.06	101.830	50.71	5.00	.09	.011	.013
14	.....	68.18	.....	.12	6.17	2.88	1.77	.....	21.97	.009	.004	.00	101.108	49.24	10.33	.09	.004	.004
15	62.22	.....	8.81	.17	.70	.44	1.06	.....	27.60	.009	.002	.....	101.011	50.41	12.98	.13	.002	.004
16	.....	71.11	.....	.41	3.60	7.98	2.65	.....	11.14	.025	.011	2.84	99.766	51.42	5.23	.32	.011	.011
17	.....	78.50	.....	.37	4.68	3.34	2.26	.....	8.28	.009	.020	2.18	99.539	56.76	3.19	.29	.020	.004
18	48.24	26.20	.....	.98	.82	1.29	.55	.....	21.96	.027	.002	.80	100.869	52.09	10.31	.76	.002	.012
19	.....	72.57	.....	.34	4.90	4.38	1.96	.....	12.98	.009	.021	2.21	99.370	52.47	6.09	.26	.021	.004
20	.....	83.53	.58	.17	2.53	2.81	1.49	.....	7.96	.009	.004	.25	99.333	60.85	3.74	.13	.004	.004
21	.....	75.60	1.07	.35	5.32	4.68	2.12	.....	13.35	.014	.052	1.56	99.216	51.88	6.27	.27	.052	.006
22	.....	90.42	1.20	.15	2.54	.78	1.02	.20	4.09	.021	.013	.....	100.434	66.31	1.92	.12	.013	.009
23	.....	65.14	1.63	.37	7.02	5.19	1.56	.....	15.15	.018	.011	3.02	99.109	48.37	7.11	.29	.011	.008
24	.....	67.59	1.12	.28	4.75	6.10	1.96	.....	15.98	.011	.023	1.28	99.094	49.75	7.50	.22	.023	.005
25	.....	62.11	.19	.36	7.08	6.25	2.33	.....	21.25	.066	.051	1.39	101.077	45.06	9.97	.28	.051	.029
26	.....	72.44	1.12	.44	6.20	4.03	1.40	.....	12.05	.021	.055	2.16	99.916	53.25	5.66	.34	.055	.009
27	.56	91.98	.....	.13	2.33	.59	.58	.79	3.35	.025	.001	.04	100.376	66.90	1.57	.10	.001	.011

[a] Analyses made at the Degerfors Iron Works laboratory.

[b] Samples taken at the works.

[c] CO_2, 7.02; C, 0.36.

TABLE 8.—*Analyses of iron, slag, and furnace gas.*

INCOMPLETE ANALYSIS OF IRON.

Date.	Number of charge.	C	Si	Mn	S	P
1911.		*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Jan. 3	138	4.19	1.35	0.90	0.004	0.021
Jan. 14	171	4.04	.75	.82	.005	.020
Mar. 16	324	3.00	.14	.08	.028	.019
Mar. 30	372	3.10	.45	.40	.014	.010

ANALYSIS OF SLAG.

Date.	Number of charge.	SiO_2	Al_2O_3	TiO_2	FeO	MnO	CaO	MgO	CaS	P_2O_5	Total.
1911.		*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
Jan. 3	138	44.76	1.20	2.58	1.75	1.35	31.70	15.10	0.077	Trace.	98.517
Jan. 14	171	41.60	6.85	2.72	1.49	1.48	28.91	16.70	.063	0.000	99.313
Mar. 16	324	46.82	5.06	.....	6.89	.23	33.27	7.97	.023	.041	100.274
Mar. 30	372	37.98	6.98	.37	1.28	.52	27.98	23.45	.123	.000	98.683

INCOMPLETE ANALYSIS OF SLAG.

Date.	Number of charge.	Fe	Si	Mn	S	P
1911.		*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Jan. 3	138	1.36	21.05	1.05	0.034	Trace.
Jan. 14	171	1.16	19.56	1.14	.028	0.000
Mar. 16	324	5.36	21.97	.18	.009	.018
Mar. 30	372	1.00	17.82	.40	.054	.000

ANALYSIS OF FURNACE GAS, BY VOLUME.

Date.	Number of charge.	CO_2	O	CO	H	CH_4	N
1911.		*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Jan. 3	138	28.2	.....	.....	.....	.....	.....
Jan. 14	171	27.2	0.0	57.5	14.8	0.0	0.5
Mar. 16	324	12.6	.....	71.9	13.0	1.7	.8
Mar. 30	372	19.2	.....	59.7	17.6	2.5	1.0

REPORT ON OPERATION OF THE FURNACE FROM AUGUST, 1911, TO MAY, 1912.

The furnace was again put into operation in August, 1911, and was in continuous operation up to July, 1912, when it was relined. In May, 1912, a report was submitted to the Jern Kontoret by the engineers in charge of the operation of the furnace from August, 1911, up to that time. The following summary is taken from an abstract of the report that appeared in Metallurgical and Chemical Engineering, July, 1912, p. 413:

Results of operation of furnace from August, 1911, to May, 1912.

Iron, in ore, per cent	60.95
Iron, in burden, per cent	66.84
Weight of slag per ton of iron, kg	328

Weight of fuel per ton of iron, kg	404
Volts on furnace	73.6
Amperes on furnace	11,423
Power on furnace, kw	1,482
Power per ton of iron, kilowatt-hours	2,225
Output of iron per kilowatt-year, tons	3.94
CO_2 in throat gas, per cent	23.49
Volume of circulating gas, cubic meters per second	0.24
Pressure of gas in furnace, mm. water	225
Temperature at bottom of shaft, °C	441
Temperature at middle of shaft, °C	279
Temperature at top of shaft, °C	17
Electrode consumption, per ton of iron:	
Gross, kg	5.72
Net, kg	5.18

The variation in the charcoal and power consumption per ton of iron produced, depending upon the iron content of the ore used, is shown in the following tabulation, taken from the article mentioned above:

Variation in power and charcoal consumption.

Fe in ore, per cent	67.66	66.71	57.15	54.83
Slag per ton of iron, kg	166	209	409	521
Coal per ton of iron, kg	347	380	439	490
Kilowatt-hours per ton of iron	1,819	2,253	2,369	2,525

The composition of the pig iron produced varied between the following limits:

Variation in composition of pig iron.

Element.	Range.		Average.
	Low.	High.	
	Per cent.	*Per cent.*	*Per cent.*
Carbon	2.688	3.859	3.406
Silicon	.183	2.473	.725
Manganese	.190	1.366	.477
Sulphur	.0087	.0841	.0126
Phosphorus	.0134	.0454	.0200

The composition of the gases produced varied as follows:

Variation in composition of gases.

Gas.	Range.		Average.
	Low.	High.	
	Per cent.	*Per cent.*	*Per cent.*
CO_2	14.63	31.80	23.49
CO	31.80	71.10	63.15
H_2	7.87	14.50	10.35
CH_4	.82	2.37	1.52
N_2 (difference)	.50	3.41	1.49

The ratio of CO to CO_2 by volume varied from 4.8 to 1.8. The calorific power of the gas per cubic meter varied from 2,035 to 2,569, an average of 2,297 calories, or from 57.5 to 72.7 calories per cubic foot.

The oxygen ratio of the slags produced varied from 2.09 to 1.23. Analysis of the slags showed the following variations in composition:

Variation in composition of slags.

	Per cent.
SiO_2	36.54 to 45.15
Al_2O_3	1.96 to 9.20
TiO_2	Trace to 7.42
FeO	.58 to 6.22
MnO	.19 to 6.00
CaO	21.08 to 45.58
MgO	6.22 to 26.65
CaS	.04 to .44
P_2O_5	Trace to .034

The temperatures of the iron and the slag issuing from the furnace varied as follows:

Iron, °C	1,230 to 1,420
Slag, °C	1,290 to 1,460

In September, 1912, the Jern-Kontoret leased the plant at Trollhättan to the Strömsnäs Iron Works, which has since worked it as a commercial plant. The final report on the research work at Trollhättan, as well as a report on similar commercial installations elsewhere, has now been published.[a] A summary of the report, as furnished by Electro-Metals, of London, is largely presented below.

Use of concentrates.—The proportion of concentrates ought not to exceed 20 per cent of the ore charged. This figure, however, does not appear to be final, as in a somewhat modified furnace of the same type constructed later at the plant of the Uddeholm Co. at Hagfors 25 per cent of concentrates is used without any difficulty.

Power consumption.—The power consumption per ton of pig iron varies in proportion to the iron content in the ore. A poor ore and a pig iron high in silicon and manganese require more power than rich ore and pig iron low in silicon and manganese. For such iron the power consumption averages only 2,067 kilowatt-hours per ton of pig iron—that is, there is obtained 4.22 tons of pig iron per kilowatt-year, or 3.10 tons per horsepower-year.

Charcoal consumption.—The charcoal consumption per ton of pig iron varies from 20 to 24 hectoliters (57 to 68 bushels), depending on the quality of the charcoal and the charge. Coke, unless mixed with charcoal, is unsuitable for this furnace.

[a] Abstract of report, Iron and Coal Trades Rev., vol. 86, 1913, p. 714.

Consumption of electrodes.—The consumption of electrodes at Trollhättan has been reduced to less than 3 kilograms per ton of pig iron. At Hagfors it has amounted to as much as 6 to 9 kilograms. This discrepancy is explained by the fact that the electrode consumption is increased in proportion to the higher power consumption for a poor charge, and is further increased by the more efficient circulation of gases and higher carbon dioxide content in the gas. The lower electric load per unit of surface at the Hagfors furnaces also contributes to the higher electrode consumption at this plant.

Cost of repairs.—The cost of repairs is lower than was at first expected. In the manufacture of pig iron containing silicon and manganese the cost for repairs is higher than in producing pig iron with a low content of those elements.

Quality of the pig iron.—The silicon content does not vary more than in the product of an ordinary blast furnace. The phosphorus content is lower than when the same quality of charge is used in an ordinary blast furnace, owing to the lower consumption of charcoal in the electric furnace. The sulphur content is, however, slightly higher in the product of the electric furnace. However, it should be observed that both at Trollhättan and at Hagfors unroasted ores have been used without any difficulty arising from the sulphur present.

The quality of the pig iron from the electric furnace has been highly commended. It acts particularly well in the open-hearth furnace, and steel made from it is certainly not inferior to steel made from ordinary pig iron. E. Odelberg, managing director of the Strömsnäs Iron Works, states that the electric pig iron is "of the very best quality for the open-hearth process and, as regards the uniformity of the silicon content, fully as uniform as iron from an ordinary blast furnace." A. Herleinus, managing director of the Uddeholm Co., states that "the electric pig iron has been used with satisfactory results, both for the open-hearth, Bessemer, and Lancashire processes. Generally speaking, there has been no difficulty in obtaining pig iron of uniform quality, although slightly better uniformity may possibly be obtained with a very carefully conducted blast furnace."

Value of the gas.—At the plant of the Uddeholm Co., at Hagfors, the gas from the furnaces has been used with very good results for heating the open-hearth furnaces. It is estimated that the value of the gas obtained per ton of pig iron may be taken at 2.50 kroner (about $0.67).

With regard to the furnace that was used, the report expresses the opinion that its construction and general dimensions have been found to be suitable. A summary of the most important figures relating to the economic results is presented in Table 4 following.

TABLE 4.—*Results of electric iron smelting with furnace at Trollhättan.*

Item.	Nov. 15, 1910, to May 29, 1911.	Aug. 4, 1911, to June 21, 1912.	Aug. 12, 1912, to Sept. 30, 1912.	October to December, 1912.
Ore, concentrates and briquets, kg	4,336,338	7,917,214	[a] 1,406,530	2,914,830
Limestone, kg	345,405	647,479	108,150	169,944
Charcoal, hectoliters	65,474.5	107,282.5	21,859.5	44,934.5
Coke, kg		70,854		
Electric energy, kw.-hours	6,339,131	10,845,180	1,939,073	3,957,565
Iron content of ore, per cent	60.79	60.75	68.67	65.38
Iron produced, kg	2,636,098	4,809,670	965,915	1,905,865
Slag per ton of iron, kg	350	324	192	
Electrodes, per ton of iron, gross kg	10.00	6.08	3.02	2.78
Electrodes per ton of iron, net kg	4.95	5.17	3.02	2.78
Charcoal per ton of iron, hectoliters	24.84	22.31	22.63	23.58
Working time, hours, minutes	4,441 20	7,218 23	1,173 8	2,158 30
Repairs, hours, minutes	236 53	506 07	13 47	49 30
Repairs, percentage of total time	5.06	6.55	1.16	2.24
Average load, kw	1,427	1,502	1,653	1,833
Kilowatt-hours, per ton of iron	2,405	2,255	2,007	2,076
Iron per kw.-year, tons	3.64	3.88	4.36	4.22
Iron per hp.-year, tons	2.68	2.86	3.20	3.10

[a] 1,209,825 kg. represented Kiiruna "A" ore containing 69.61 per cent of Fe.

The table shows that step by step the results have been improved, the quantity of iron per horsepower-year increased, whereas the electrode consumption and time for repairs have been reduced, these three items in conjunction with the saving in charcoal being the decisive factors as regards electric iron smelting.

The figures show that during the last few months of the experimental work as much as 3.20 to 3.10 tons of iron were obtained per horsepower-year, whereas before the alterations were made the highest average figure was 2.86 tons. The highest expected yield was 3 tons per horsepower-year, which was therefore exceeded. It may also be mentioned that during single periods of several weeks when especially suitable ores were used, the highest average figures above mentioned were materially exceeded. It will thus be seen that as regards efficiency the results of the furnace surpassed expectations. The electrode consumption shows a remarkable decrease. The consumption of 13.8 kilograms during the first period of working was finally reduced to about 3 kilograms per ton of pig iron. The cost and time required for repairs were items that could not be estimated beforehand. Experience has shown that both the cost and the time required are less than could have been expected. Utilization of the gas, when practicable, should also be taken into account. At Hagfors the use of gas for firing the open-hearth furnaces is estimated to reduce the cost of the pig iron about 65 cents per ton.

DURABILITY OF THE ROOF OF THE CRUCIBLE.

In perfecting the electric furnace one of the most serious difficulties that had to be overcome was maintaining the brickwork of the crucible. In the operation of the furnace at Trollhättan from August 4, 1911, to June 21, 1912, the following repairs were made: October 26, 1911, the arch over the crucible was partly repaired;

February 13, 1912, the arch was entirely rebuilt; April 27, 1912, the arch was partly repaired. At Hagfors the arch was rebuilt after working four and one-half months.

COMMERCIAL FURNACES OF THE SWEDISH TYPE.

As a result of the successful working of the furnace at Trollhättan the following furnaces have been built:

Furnaces built subsequent to the Trollhättan furnace.

Place.	Number of furnaces.	Horsepower of each.	Total horsepower.
Sweden:			
Domnarfvet	1	3,500	3,500
Hagfors	2	3,000	6,000
Norway:			
Hardanger	2	3,500	7,000
Arendal	3	3,000	9,000
Switzerland	1	2,500	2,500

However, the furnaces at Hardanger, after having been operated for about nine months, were closed down, the reason given being that suitable raw materials were not available in Norway. In Sweden charcoal is used as a reducing agent, whereas in Norway only coke is available for that purpose. During the experimental work at Trollhättan, coke was tried as a reducing agent, but was found unsatisfactory, and the failure at Hardanger, where coke only was used, confirms the conclusions reached by the engineers at Trollhättan regarding the unsatisfactory results with the use of coke as a reducing agent.

The bus-bar connections and the arrangements of electrodes around the crucible in the Swedish type of electric shaft furnace are shown in figure 13.

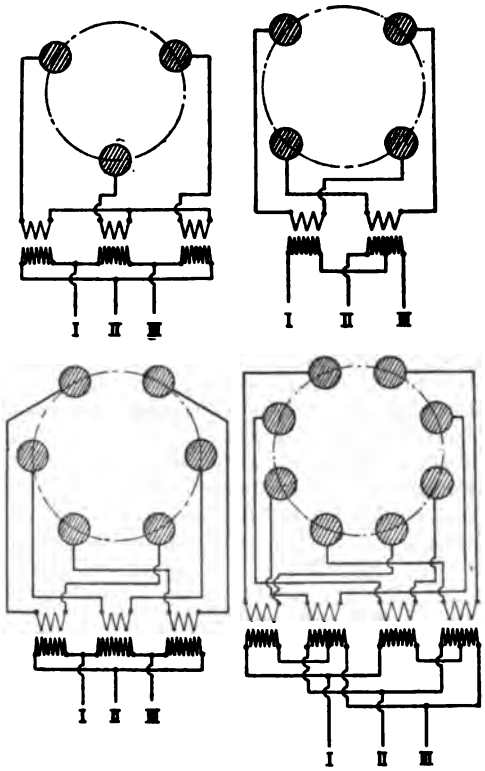


FIGURE 13.—Bus-bar connections and arrangement of electrodes in Swedish type of electric shaft furnace.

The Uddeholm Co., at Hagfors, Sweden, is adding a third furnace to its electric furnace installation at Hagfors, and is constructing

three similar furnaces at its Nykroppa works. It will probably substitute electric smelting altogether for its blast furnaces.

The Stora Kopparbergs Bergslags is also preparing to introduce electric smelting on a large scale, and according to reports, other iron works in Norway and Sweden will follow its example.

The cost of construction of the plant at Trollhättan is given below:

Cost of construction of electric iron smelting plant at Trollhättan, Sweden.

Preparation of site:		
Excavations, foundations, and fencing; narrow-gage tram lines, 3 turntables, and 5 trucks; 1 5-ton and 1 40-ton weigh bridge; railroad siding (about 1,830 feet) with 1 turntable; earthenware water main from the canal (400 feet)		$10,727.42
Buildings:		
Furnace house with 2 windlasses and inclined tracks	$14,735.00	
Charcoal storage house with conveyors and elevators	6,032.51	
Crusher house and elevator	1,282.97	
Office, laboratory, and storeroom	1,393.87	
Repair shop with equipment	1,040.63	
General storehouse	243.64	
		24,728.62
Furnaces:		
1 furnace for 2,500 horsepower (ironwork, castings, masonry, etc.)	13,117.12	
Electric equipment	13,781.70	
Bars and cables to the electrodes	3,381.93	
Exhaust fan for furnace gases	826.94	
Pump and motor (for water)	1,452.64	
Water tank and pipes	942.19	
		33,502.52
Crusher		1,010.80
Electric lighting and motors: Transformers and wiring; 4 motors for exhaust fan, crusher, ore and coal elevators, and conveyors; direct-current transformers		5,157.24
Laboratory equipment		889.88
Instruments (self-recording) for measuring temperature, pressure, and volume of gas		1,894.25
Tools and sundry supplies		1,212.30
Furniture and fittings for office, etc		675.00
Supervision and sundry expenses		5,637.86
Drawings, specifications, and license fee		7,500.00
Total		92,985.89

CHEMISTRY OF THE REDUCTION OF IRON ORES IN THE ELECTRIC FURNACE.

As is well known, the reduction of iron ores is effected by heating the ore and the reducing agent to such a temperature that a reaction takes place between the oxygen and the reducing agent, which usually is some form of carbon. For the most part, iron ores at the present time are reduced in the blast furnace. In order to get a clear idea of the difference between a blast furnace and an electric-reduction

furnace, let us first briefly note how the work of reduction and subsequent melting of the reduced iron and fluxes is brought about in the blast furnace.

The charge is composed of ore, oxide of iron plus gangue materials, fluxes for combining with the gangue materials of the ore in such proportions determined by analysis and calculation as will make a fusible slag, and the fuel, which is needed to provide heat and also acts as a reducing agent. This fuel is generally coke or charcoal and contains practically no volatile matter. In the tuyère zone the larger part of the fixed carbon passes to CO, but whether it first passes through the CO_2 state and is then reduced to CO by incandescent carbon is still a debated subject.[a] At any rate CO is the final product, and in its production a high enough temperature is produced to melt down the previously reduced iron and to form a fluid slag from the fluxes and gangue materials present. At this point the difference between the blast furnace and an electric-reduction furnace should be noted, for in the electric furnace the heat necessary for melting the charge is furnished by the electric current. In the blast furnace the quantity of fuel necessary to produce the smelting temperature, and not the amount necessary for the reduction of the oxides, determines the amount of fuel that is used. In the electric furnace the amount of electric energy necessary for producing the smelting temperature is used in place of coke or charcoal; and as only one-third or less of the coke that is used in blast-furnace work is needed in the electric furnace for the reduction of the iron oxides, it is possible to produce in the electric furnace three times as much iron with 1 ton of coke or charcoal as in a blast furnace. Such being the case, the amount of electrical energy and carbon that are necessary to produce 1 ton of pig iron in an electric furnace may now be considered.

This problem has been carefully worked out by Yngstrom,[b] Richards,[c] and others, and hence it is necessary in this connection to give only the essential details in such a calculation.

In determining the amount of electrical energy necessary to produce 1 ton of pig iron from a given ore the amount of heat absorbed by the following processes must be calculated: The reduction of the iron oxides to iron; the reduction of the SiO_2 to Si; the melting and superheating of n kilograms of iron; the melting and superheating of n kilograms of slag; the heating of x kilograms of CO_2 plus CO to whatever temperature it is determined that they should escape from the shaft.

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[a] Belden, A. W., Foundry-cupola gases and temperatures: Bull. 54, Bureau of Mines, 1913, pp. 14–19.

[b] Yngstrom, Lars, Electric production of iron from iron ore at Domnarfvet, Sweden: Engineer (London), Feb. 25, 1910, p. 206.

[c] Richards, J. W., Metallurgical calculations, pt. 2, 1907, p. 403.

From the number of calories absorbed in this manner may be deducted the heat that would be developed by the combustion of the x kilograms of carbon needed for the reduction. The calculation of the amount of carbon needed for the reduction of the oxide is not an easy matter, for, as has been pointed out by Richards, in calculating the amount of carbon necessary to reduce the oxide of iron there is no way of knowing what proportion of carbon will form CO and what proportion CO_2. The amount of each that will be formed depends upon the temperature, CO being almost the only product that is formed at a high temperature, whereas CO_2 is formed in increasing quantities as the temperature decreases, the reactions being represented as follows:

$$Fe_2O_3+3C = Fe_2+3CO$$
$$Fe_2O_3+2C = Fe_2+CO+CO_2$$

From these equations we note that if the gaseous product of the reduction is all CO, not more than one-third as much carbon is required for the reduction as is required in the case of blast-furnace practice. Yngstrom in his calculations assumes that when carbon combines with oxygen in the electric furnace a mixture is formed that contains 30 per cent (by volume) of CO_2, and on this basis he calculates the carbon necessary for the reduction of the iron oxides present and the subsequent development of heat by the formation of CO and CO_2 by the following equation:

$$14Fe_2O_3+30C = 12CO_2+18CO+28Fe$$

In his computations he uses the following theoretical values:

To reduce 1 kg. of Fe from Fe_3O_4 there is required 1,650 calories.
To reduce 1 kg. of Fe from Fe_2O_3 there is required 1,800 calories.
To reduce 1 kg. of Si from SiO_2 there is required 7,830 calories.
By oxidation of 1 kg. of C to CO_2 there is developed 8,080 calories.
By oxidation of 1 kg. of C to CO there is developed 2,470 calories.
1 kilowatt-hour corresponds to 857 calories.
1 kg. of pig iron requries for melting and superheating 280 calories.
1 kg. of slag (monosilicates) requires 595 calories.

As a result of his calculations it is seen that for the production of 1 ton of pig iron containing 3 per cent of carbon, 1 per cent of silicon, 96 per cent of iron, and traces of manganese, phosphorus, and sulphur, made from a charge containing 60 per cent iron in the form of Fe_3O_4 and with the escaping gases containing 30 per cent of CO_2, 248 kg. of carbon, or 292 kg. of coke, and 1,460 kilowatt-hours would be required, which would correspond to 4.4 tons of pig iron per horsepower year of 365 days.

The production of this amount of iron per kilowatt-year has not been attained in actual practice, for, according to the latest reports from Trollhättan, only 3.94 tons is produced per kilowatt-year. At this point it may be interesting to compare the number of heat

calories computed by Yngstrom as being necessary to produce 1 ton of pig iron with the number that can be calculated from Junge-Hermsdorf's "Heat Balance of a Blast Furnace of 250-ton Capacity." A furnace producing 250 tons in 24 hours will produce, in round numbers, 10 tons an hour, and as 10 tons of coke is needed per hour, this would practically correspond to 1 ton of coke for each ton of iron produced.

One ton of coke is equivalent to 1,000 kg., and as 1.18 kg. of coke is equivalent to 1 kg. of pure carbon, 1 ton of coke is equivalent to practically 850 kg. of pure carbon. If 4,153 kilogram-calories, as estimated by Yngstrom, be taken as the number of calories that are given off when 1 kg. of carbon combines with oxygen, forming a mixture that contains 30 per cent (by volume) of CO_2 ($0.3\times8{,}080+0.7\times2{,}470=4{,}153$), the heat necessary for producing 1 ton of pig iron in the blast furnace is $850\times4{,}153 = 3{,}530{,}050$ calories. As 2,121,284 calories represents the amount of heat that is necessary to produce 1 ton of pig in the electric furnace, there is a difference of 1,408,766 calories between the heat necessary for producing 1 ton of pig iron in the blast furnace and that required for producing 1 ton of pig iron in the electric furnace.

PROBLEMS IN THE ELECTRIC SMELTING OF IRON ORES.

Having thus traced the history and evolution of the electric pig-iron furnace up to the present time, and having stated the fundamental chemical principles upon which the reduction of iron is based, the authors will now briefly consider some of the difficulties that had to be overcome in order to bring the Swedish type of furnace up to its present stage of development and some of the problems that yet remain to be solved.

EARLY DIFFICULTIES.

An inspection of Harmet's drawings, and of the drawings of practically all others who gave their attention to the development of an electric furnace for the reduction of iron ores, shows that the first idea was to construct a shaft similar to a blast-furnace shaft, and then to substitute electrodes for tuyères. This seemed feasible, but in practice the furnace wall in the neighborhood of the electrodes proved to be short lived. Water cooling did not obviate this difficulty, but rather tended to increase it, due to jackets burning out and to other reasons. As was early observed by Héroult, the proper way to maintain the walls of an electric furnace crucible is to remove the electrodes as far as possible from the side walls. In the development of the furnace in California and in Sweden it was found necessary not only to do this, but also to keep the charge as far as possible

from the roof of the crucible, as otherwise the roof was short lived owing to the intense local heat generated at the point where the electrodes enter the charge. In overcoming this most serious difficulty the present shape of the roof of the crucible and the manner in which the electrodes are introduced into the crucible have been evolved. As a further protection to the roof of the crucible, a part of the gases that were escaping from the top of the shaft, as previously stated, was returned to the crucible in order to cool the walls, as well as to assist in the reduction of the charge.

ELECTRODE PROBLEM.

Although the Swedish experimenters did not have so much trouble with electrodes, in California the difficulty of procuring suitable electrodes greatly interfered with the progress of the work. When experiments were begun in California the manufacturers of electrodes in this country had not previously been required to furnish electrodes of such large size, namely, about 20 inches square and about 72 inches long. Of those first furnished, that part of the electrode projecting into the crucible would either break off completely after it became heated, or else would spall off in large chunks and give trouble in operating the furnace. That others also encountered this difficulty is evident from the following quotation from a paper[a] presented by W. R. Walker at the April, 1912, meeting of the American Iron and Steel Institute:

> Our problems—mechanical, metallurgical, and otherwise—proved many, and our experience soon demonstrated that the conditions surrounding the successful operation of a large electric furnace were in many respects entirely different from those involved in the use of smaller units. In illustration the demands of a 15-ton electric furnace proved to be far in advance of the art of manufacturing electrodes. Our necessities represented a requirement that the electrode manufacturers of America and Europe had not been called upon to meet, and it took much time and money before there was finally accomplished the 20-inch round amorphous-carbon electrode that is now being used at South Chicago.

At the plant of the Noble Electric Steel Co. it was finally decided to try graphite electrodes. These worked satisfactorily in all respects except that on account of the angle at which it was necessary to insert them in the crucible they were subjected to a severe strain which caused them to break at the threaded joints. The electrode problem is, however, no longer a serious matter. As stated by Walker, the 20-inch round electrodes now in use at South Chicago give satisfaction, and the engineers at Trollhättan also report that the large carbon electrodes of about the same size as those used at South Chicago meet their requirements.

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[a] Walker, W. R., Electric furnace as a possible means of producing an improved quality of steel: Metall. Chem. Eng., vol. 10, 1912, p. 371.

UNSOLVED PROBLEMS.

In connection with the unsolved problems in the electric smelting of iron ores it may be well to take up first a discussion of the results obtained at Trollhättan and in California.

A most careful record was kept of the work done at Trollhättan during the period November 15, 1910, to April, 1911, and the data thus obtained were incorporated in a report submitted to Jern-Kontoret by two of its engineers—J. A. Leffler and E. Odelberg. The essential parts of the report have been translated by Dellwik.[a] This report has been ably discussed by Robertson[b] in a paper that he presented before the Toronto meeting of the American Electrochemical Society, September, 1911. The work at Trollhättan is also the subject of a paper by Frick.[c] In this paper Frick also discusses the results obtained in California. Some of the points brought out in his paper are discussed below.

VOLUME OF SHAFT AS COMPARED WITH CAPACITY OF FURNACE.

Theoretically, the volume of the shaft of an electric furnace should be great enough to permit the practically complete reduction of the ore before it enters the crucible. In other words, the ideal condition in electric reduction furnace work would be to have the charge in such a condition as to require only melting by the time it comes into proximity with the electrodes. So far the degree of reduction that has taken place in the shaft of the electric furnace has varied all the way from nothing up to a considerable proportion less than complete reduction. Granted that the size of the shaft at Trollhättan has been properly calculated, and that the ratio of the volume of charge in 24 hours to the volume of the furnace should be 1:55, then in order to insure the best economical working conditions of the furnace, the reduction of the charge in the shaft must take place in the proper manner.

REDUCTION OF ORE IN SHAFT.

In an ordinary blast furnace the weight of the gases produced exceeds the weight of the charge by 30 to 50 per cent, whereas in electric-furnace reduction the gases evolved amount to only about 40 per cent, by weight, of the charge. In other words, in the blast furnace three to four times as much gas is given off in the production of 1 ton of iron as is given off in the production of 1 ton of iron in the electric furnace. Moreover, the temperature of the gas as it leaves the vicinity of the tuyères may be as high as 1,600° C., whereas the

[a] Dellwik, C., Electric iron smelting at Trollhättan, Sweden: Iron and Coal Trades Rev., vol. 82, June 9, 1911, pp. 957–962.

[b] Robertson, T. D., Recent progress in electrical iron smelting in Sweden: Trans. Am. Electrochem. Soc., vol. 20, 1911, p. 375.

[c] Frick, Otto, Electric reduction of iron ores with special reference to results obtained in Electro-Metals furnace at Trollhättan, Sweden, and Noble furnace at Heroult, Cal.: Metall. Chem. Eng., vol. 9, Dec., 1911, p. 681.

highest temperature stated to have been attained at the lower end of the stack in the work at Trollhättan was 965°. Granting that the temperature of the gas as it enters the stack may be even 1,000°, inasmuch as the weight of the gases produced is only about 40 per cent of the weight of the charge, the charge would not be heated to more than about 350° C., as pointed out by Frick, owing to the fact that the specific heat of the gas and that of the charge are about the same. For this reason complete reduction in the shaft can not be accomplished solely by the heat from the gases generated in the regular manner. In other words, the shaft acts merely as a preheater, most of the reduction taking place in the crucible by means of solid carbon.

Thus it is seen that an electric furnace may be operated in one of two ways: The ore may be reduced in the crucible by means of solid carbon, with no attempt at reduction in the stack, as is done in Cal-

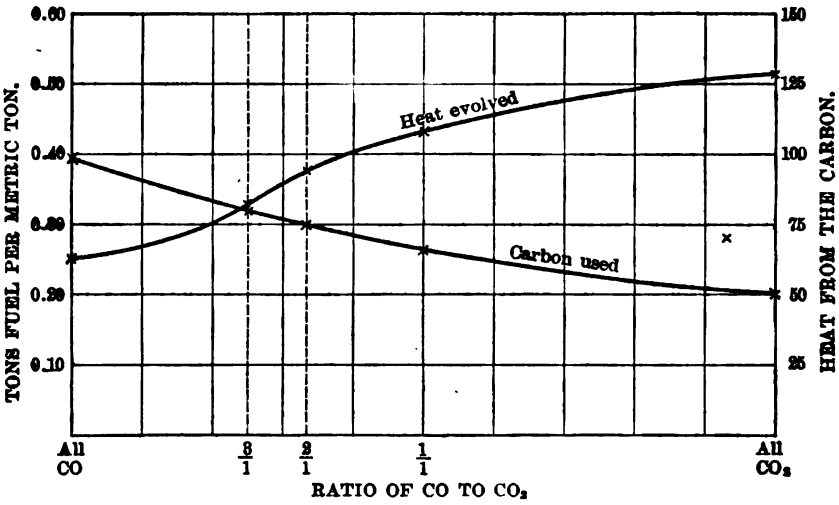

FIGURE 14.—Curves showing variation of fuel required and of heat evolved.

ifornia at the present time by the Noble Electric Steel Co., or there may be part or complete reduction in the stack, this being the principle on which the Swedish furnaces are operated.

As is well known, a method that may seem best theoretically is not always best in practice. In this instance it is not possible to judge by results, as sufficient data are not available, and so the matter can be viewed only theoretically. In the reduction of any given iron ore, a definite amount of oxygen must be removed for every ton of iron obtained. Therefore it would theoretically be best to remove all the oxygen in the form of CO. However, as the oxidizing action of CO_2 gas varies with different temperatures, such removal is not possible, as experiments show that the ratio of CO_2 to CO should not be greater than 1:1, whereas 1:2 is the determined ratio in which they are usually present in blast-furnace gases. The curves in figure 14,

the work of Prof. Richards, show that in electric-furnace work the greater the proportion of CO_2 in the gas the less is the amount of carbon required to remove the oxygen from the ore. Figure 15 (also prepared by Richards) shows that the consumption of electrical energy is proportional to the production of CO. Therefore, judging from deductions made from the curves, the efficiency of the electric furnace will be increased as reduction is effected by the gases in the shaft, and the most desirable ratio of CO_2 to CO in the escaping gases is 1:1. If this be granted, it would seem that reduction in the shaft is necessary, and the problem becomes how to effect the most satisfactory reduction in the shaft.

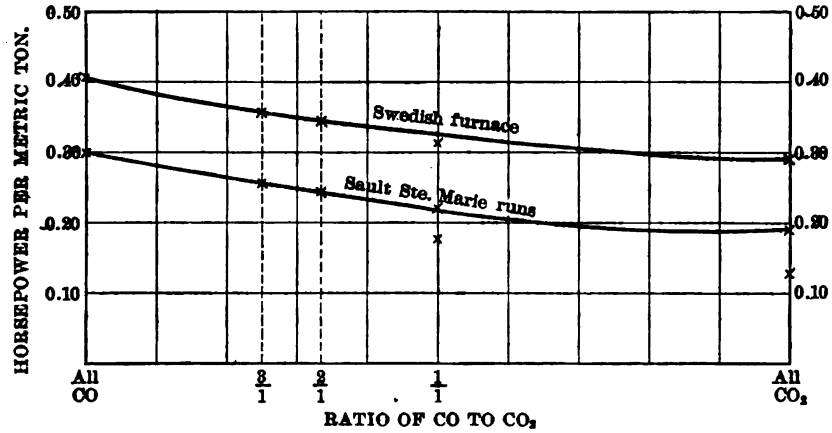

FIGURE 15.—Curves showing variation of power required according to the composition of the waste gases.

The essential factors in accomplishing reduction in the shaft are temperature, time, and the proportion of CO in the gas.

As has just been pointed out, the volume of the gas generated by the reduction of the oxides is not sufficient to maintain a reduction temperature in the stack. As a remedy for this difficulty the present system of gas circulation was adopted. The system is briefly discussed below.

GAS CIRCULATION.

As has been pointed out by Frick,[a] the idea of using circulating gas in the electric furnace was conceived by Harmet and incorporated by him in his papers[b] on the reduction of iron ores in the electric furnace.

[a] Frick, Otto, Electric reduction of iron ores, with special reference to results obtained in Electro-Metals furnace at Trollhättan, Sweden, and Noble furnace at Heroult, Cal.: Metall. Chem. Eng., vol. 9, 1911, p. 631.

[b] Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in Europe: Mines Branch, Department of the Interior, Canada. 1904, pp. 124, 139.

METHOD OF UTILIZATION.

At present gas circulation is used in the Swedish type of furnace in order to cool the superheated brickwork of the crucible and to carry up through the stack a volume of gas that is large enough and hot enough to bring about reduction in the stack.

This method has been the subject of a great deal of discussion.

OBJECTIONS.

The following are the principal objections to the circulation of gas in the electric furnace:

(1) The moisture of the charge, the CO_2 of the flux, and the CO_2 gas naturally produced in the furnace are returned in a large part to the crucible and react on the unconsumed carbon by the reactions—

$$CO_2+C=2CO$$
$$H_2O+C=H_2+CO.$$

(2) The reactions noted materially cool the smelting zone of the furnace, which is its most vital working part.

(3) The system is cumbersome and expensive, and soon reaches the maximum of useful effect.

In order to overcome the objections mentioned it has been proposed to keep the ore in the shaft at a low red heat, and then to allow the CO gas produced by reduction in the crucible to rise slowly up through the charge, thus giving it the best opportunity of producing the maximum amount of CO_2. In the circulation of gas, as has been pointed out by Richards,[a] it is not the increased volume of the circulating gas that performs the reduction, as it is evident that when the amount of gas passing through the furnace is increased two or three times its velocity is also increased to that extent, and therefore its contact with the ore is only one-half to one-third as long. Such being the case, the only value of circulating a part of the gas escaping from the top of the shaft, so far as the problem of reduction is concerned, is that the charge in the shaft is kept at the temperature necessary for reduction, but, on the other hand, the CO_2 returned to the crucible is reduced by C to CO, and thus one aim of electric-furnace reduction is defeated, namely, the removal of the oxygen from the ore with a minimum amount of carbon.

Nevertheless there are objections to the methods that have been proposed for obviating the difficulties connected with the present method of gas circulation which may be briefly stated as follows:

(1) It would probably be difficult to heat successfully the charge in the stack by means of surface electrodes embedded in the walls of the shaft.

(2) As at a low red heat (say 480° C.) reduction takes place slowly, the volume in the shaft would have to be large in order to

[a] Richards, J. W., Gas circulation in electrical reduction furnaces: Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 403.

give the charge time enough to be reduced in passing through the shaft.

(3) The use of burnt limestone, where shaft reduction is attempted, would introduce an unnecessarily large quantity of fine material into the charge.

(4) The water-cooled plates in the arch of the crucible, although useful, would not of themselves form a satisfactory substitute for the introduction of cold gas beneath the roof of the crucible.

As has been stated by Leffler,[a] one of the engineers of the Jern-Kontoret, gas circulation would be abandoned for a more simple and practicable method if such should be discovered.

However, the use of gas circulation at Trollhättan seems to be necessary, at least with the Swedish type of furnace. If this be true, efforts to improve the system ought to be directed somewhat along the following lines:

As is now done at Trollhättan, that part of the gas from the top of the shaft that is to be circulated could be drawn through a "Cottrell system" for the removal of the solid particles of ore and flux, by means of the circulating fan, and forced through a regenerator filled with coke heated electrically to such a temperature as to bring about the reduction of the CO_2 to CO. The gas at the same time would be heated, but not to a temperature too high to absorb heat as it entered the crucible, and would thus pass up into the shaft at such a temperature as to readily effect reduction. Part of the excess waste gases could then be burned in a jacket around the shaft of the furnace so as to prevent radiation losses; the waste gases from the jacket could be passed to a preheater and completely burned, whereby the charge, minus the reducing agent, would be dried and raised to a temperature high enough to admit of the ore being reduced as soon as or soon after it was introduced into the stack.

The obvious advantages of such a system would be the doing away with the introduction of water vapor into the gas, as is done by the present system of scrubbing the gas; the removal of CO_2 from the gas; and the raising of the temperature of the circulating gas.

Although the system as above outlined is just as cumbersome and expensive as the present method, it would probably be more effective.

OTHER METHODS SUGGESTED.

Some of the methods suggested as improvements are:

(1) *Calcining the limestone outside of the furnace.*—This subject is discussed in a later section dealing with the comparison of the Swedish and Californian types of furnaces.

[a] Leffler, J. A., In discussion of Richards's paper on "Gas Circulation in Electrical Reduction Furnaces": Trans. Am. Electrochem. Soc., vol. 21, 1912, p. 412.

(2) *Preheating the ore.*—Judging by experience gained in California, the writer believes that the preheating of the charge is beneficial, for the following reasons: (*a*) It dries the charge and thus permits a more accurate weighing of the same, which is especially important in the electric reduction furnaces; (*b*) the initial temperature of the charge on entering the stack is thus sufficiently high to permit its being reduced immediately.

(3) *Smelting of fine ores.*—The authors are privileged to quote from a communication from Electro-Metals, Ltd., in regard to this matter, as follows:

> As was explained in our first publication on this subject, the object of this plant [at Trollhättan] was to determine the relative merits of electric smelting as compared with ordinary blast-furnace smelting. For this reason the work has been carried out under widely varying conditions and with different kinds of ore and fuel. In consequence the results are by no means uniform and scarcely suitable for conclusions based on the average figures.
>
> One object of some importance in Sweden was to determine the proportion of ore concentrates which could be used. The results prove that a large proportion of concentrates is detrimental to smooth running and good results.
>
> This is readily understood from the fact that only about one-third as much charcoal is used as in the blast furnace, and concentrates therefore have an increased tendency to choke the passage of the gas.

As will be seen from the above, it is not feasible to use a large proportion of fine material in making up the charge for the electric furnace. This limitation, however, does not prevent the smelting of fine ores, for they may be sintered.

(4) *Sintering.*—The advantages derived from sintering in electric-furnace work would be as follows: (*a*) The fine material would be caked in lumps, permitting free passage of the gases up through the charge in the shaft, and as the lumps would be porous they would be readily reduced, and (*b*) the fine ore would be preheated; that is, the hot sintered ore could be charged directly into the shaft at reduction temperature.

(5) *Increasing the size of the unit.*—The electric reduction furnaces now in operation vary in size as regards their horsepower from 1,500 up to 3,500. The largest yet designed requires 7,500 kilowatts and is at a plant in Sweden. As can be readily understood, it is important that the size of the unit be made as large as possible, and it is quite probable that larger furnaces will be built.

(6) *Increasing the general efficiency of the furnace.*—Improvements will probably be along the following lines: (*a*) The utilization of the waste gases; (*b*) the securing of a high-power factor; (*c*) the correction of induction losses; and (*d*) the perfecting of the single-phase furnace as recommended by Catani.[a]

[a] Catani, R., Large electric furnaces in the electrometallurgy or iron and steel: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 168.

STATUS OF THE IRON INDUSTRY IN THE WESTERN STATES.

At the presenti time the only important iron-reduction plant west of the Mississippi River is at Pueblo, Colo. The plant has six stacks, twelve 50-ton open-hearth furnaces, two 15-ton Bessemer converters, four 400-ton furnaces, and two 250-ton furnaces. That there are no other large iron-reduction plants in the West is due largely to the fact that there is lacking in that part of the country, especially in California, the grade of coal necessary for making a suitable metallurgical coke. Then, too, although there are several known large iron-ore deposits scattered throughout the Western States, not much attention has in the past been paid to this class of deposits.

For nearly half a century after the discovery of gold in California the prospector along the Pacific coast looked for gold-bearing rock only. In recent years the discovery of valuable deposits of copper in California gave the prospector the copper craze, so to speak. In prospecting he looked for iron only as an indication of the presence of copper; that is, if he came across an iron deposit he looked upon it as a gossan or capping, and at once examined it for gold and copper, as his experience had taught him that gold is frequently found near the surface in rock of similar appearance and copper at a greater depth. It is well known that there are many deposits that yield oxidized ores above water level and sulphides below, so it is easy to understand why the prospector should have mistaken a true iron deposit for a deposit of the nature just mentioned. Even such iron ores as those in Shasta County were never seriously considered as such by prospectors. The miners in that part of the country commonly believe that iron indicates the presence of copper, and they insist that copper will be found below the iron.

IRON-ORE DEPOSITS ON THE PACIFIC COAST.

As above stated, it has long been known that there are large deposits of iron ore in Nevada, Arizona, and California, especially in southern California. As has been pointed out by Jones,[a] most of the present known deposits of the Southwest have been discovered in places where there has been more or less erosion. Owing to the erosion, faces of ore several hundred feet in height, and of as great or greater width, are shown in certain intersecting gulches. Intrusions in limestone beds are common in the desert regions of Arizona, western Nevada, and California, and, as Jones states, there is in these regions scarcely a range of this nature that does not show some ledges of iron or float ore, and he ventures the assertion that syste-

[a] Jones, C. C., Iron ores of the southwest: Paper read before the American Mining Congress, September, 1910; published in Min. and Min., vol. 31, April, 1911, p. 574.

matic prospecting and mining would uncover as great bodies of ore as have been already accidentally exposed by erosion.

For some time past the railway and oil interests of California have been systematically acquiring iron-ore holdings. For example, Jones[a] states that in 1911 the Union Oil Co., through its subsidiary company, the California Industrial Co., was reported to hold an aggregate proven tonnage of 300,000,000 tons, one-third being in California and two-thirds in Lower California, Mexico. This company has acquired these properties for the reason that they believe the problem of using oil as a reducing agent will ultimately be solved, and thus permit the establishment of an iron and steel industry in that section of the country. The Southern Pacific Railroad Co., operating under the name of the Iron Chief Mining Co., controls several deposits throughout the State, one of which is in Riverside County, some 140 miles from Los Angeles. Jones[b] states that it has some 30,000,000 tons of proven ore of such a nature that it can be mined cheaply. The average analysis of the ore is claimed to show 64 per cent iron and phosphorus within the Bessemer limit.

In addition to these deposits Jones also mentioned the following: At Scotts Siding, 190 miles east of Los Angeles, where 10,000,000 tons of ore have been blocked out, there being three times that quantity of ore on the property; in the Providence Mountains, 236 miles from Los Angeles, is a deposit of 5,000,000 tons of soft hematite, of Bessemer quality, that can be mined with a steam shovel; 12 miles west of Silver Lake Station and 230 miles east of Los Angeles is a deposit owned by the Colorado Fuel & Iron Co. that it states shows over 13,000,000 tons of ore that can be mined with steam shovels. In addition to the deposits named there are the Minarets deposits in Madera County, which are at present inaccessible, being about 80 miles from the railroad, but it is thought that a railroad line will soon be built into this district, thus making them accessible both to San Francisco and Los Angeles. It is stated that this is one of the largest deposits of iron ore to be found anywhere in the United States. As a result of his investigation Jones is of the opinion that there is 200,000,000 tons of available high-grade ore in southern California alone within 300 miles of Los Angeles, and that the ore can be laid down at Los Angeles at a cost of $3.50 to $4 per ton. If these figures are correct, there is a basis for the hope that an iron industry will be established on the Pacific coast, provided there can be found a commercially feasible process that will permit the utilization for metallurgical purposes of those fuels that are available in that part of the country.

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[a] Jones, C. C., Op. cit., p. 53. [b] Jones, C. C., Idem.

DEVELOPMENT OF THE ELECTRIC REDUCTION FURNACE IN CALIFORNIA.

ORE DEPOSIT OF SHASTA IRON CO.

For 30 years or more a company known as the Shasta Iron Co. has owned a deposit of iron ore situated about 7 miles from the mouth of the Pitt River, in Shasta County, Cal. This ore is a high-grade magnetite. An average of a large number of analyses gives the following percentages:

Average of analyses of magnetite from Shasta County, Cal.

	Per cent.
Fe_3O_4	89.4
Fe_2O_3	7.3
Fe	69.90
MgO	.10
MnO	.18
SiO_2	2.40
P	.011
S	.009

The ore is found near a contact between limestone and diorite. The ore body is markedly uniform, a series of 20 samples taken by the writer over a cut 60 by 40 feet giving the following iron content:

	Per cent.
Maximum	70.5
Minimum	68.8
Mean	69.7

The limestone is also of an excellent quality; an average analysis is as follows:

Average of analyses of limestone from Shasta County, Cal.

	Per cent.
SiO_2	1.2
Al_2O_3	.5
MgO	1.1
CaO	53.8
FeO	.2

The percentage of CaO is equivalent to about 98 per cent calcium carbonate.

Although the Shasta Iron Co. had previously planned to make pig iron from this ore, nothing definite was done until the summer of 1906. At this time the possibility of smelting the ore by electricity was brought to the attention of H. H. Noble, president of the Northern California Power Co. Owing to Héroult's connection with the experimental work that had been done at Sault Ste. Marie, he was commissioned to construct a plant for the purpose of treating the ores above mentioned. The place selected was at a point on Pitt River, near the mines. The place was named Heroult, and is still known by that name.

FIRST EXPERIMENTAL ELECTRIC REDUCTION FURNACE.

In July, 1907, the first furnace having been completed, experimental work was begun. This furnace was a 1,500-kilowatt, three-phase furnace of the resistance type. It soon presented mechanical difficulties that made its commercial use impracticable, and so it was closed. This furnace is shown in figure 16. In construction the crucible resembled a long, elliptical-shaped iron box. The bottom was made of heavy cast-iron plates, into which had been cast cast-iron lugs or pins, and upon the iron plates was tamped a bottom made of carbon paste. The bottom plates were connected with the bus bars in such a manner as to form a neutral in the three-phase system.

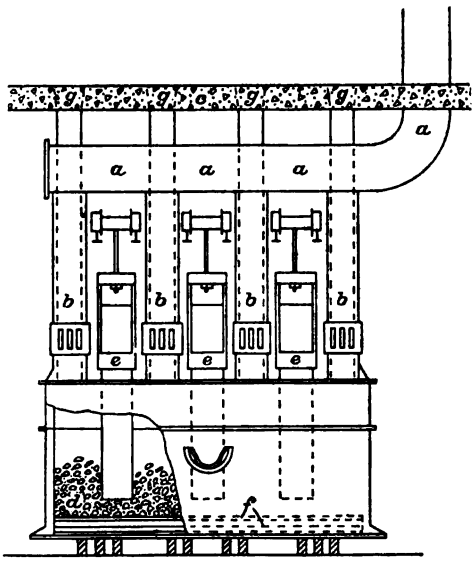

FIGURE 16.—Héroult 1,500-kilowatt, three-phase reduction furnace.

The electrodes passed through the roof, as shown in the figure. They were suspended from copper holders which were water jacketed. The charge was fed into the crucible through the pipes, *b*, and the gases from the crucible passed up around *b* into *a* to preheat the charge in *b* by burning the gases around *b*, air being admitted for that purpose through slots in the outer pipe just above the roof of the crucible. The pipes, however, proved too small, and the charge in them became so hot as to hang, and so they had to be discarded. It was also impossible to maintain a roof over the crucible, and so the furnace was finally operated as an open-top furnace. Several tons of excellent foundry iron was made in this furnace, and later it was used for making ferrosilicon, but was finally closed and removed in order to make room for the type of furnace subsequently adopted.

SECOND AND THIRD EXPERIMENTAL FURNACES.

After the closing of this furnace experimental work was conducted in a 160-kilowatt single-phase furnace, and, using the results obtained in this furnace and its method of operation as a basis, a second 1,500-kilowatt three-phase furnace was designed and built. This

furnace is shown in figure 17. Above the crucible was a superposed shaft resembling that of an ordinary blast furnace.

In the operation of the furnace the ore, mixed with the proper proportion of flux, was fed into a preheater, *b*, that was heated by waste gases from the combustion chamber *k* at the top of the shaft. The gases entered the base of the preheater through a flue, *e*, from an annular chamber surrounding the top of the shaft and communicating with the chamber *k* through openings.

A mechanically operated scale car, *g*, moved on a circular track around the top of the stack and received first a weighed charge of ore and flux from the preheater *b* and then a weighed charge of charcoal from the charcoal hopper *c*, delivering the charges alternately into the furnace.

The charge was kept at about the height of the irregular line shown in the shaft. Above the charge were small openings in the shaft with suitable valves for admitting the requisite amount of air to burn the gases resulting from the reduction of the ore in the lower part of the stack, thus still further heating the charge; as above stated, the waste gases were then passed up through the preheater *b*, thus preheating and drying the ore before it was charged into the furnace.

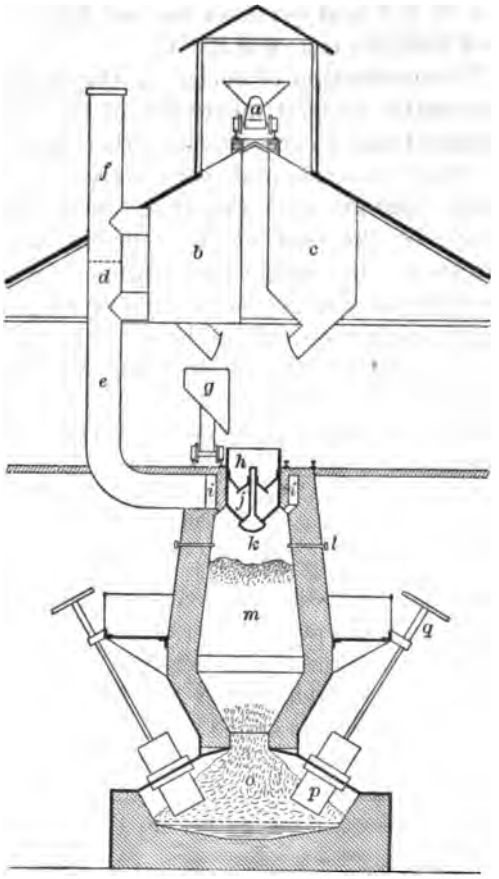

FIGURE 17.—Second 1,500-kilowatt, three-phase reduction furnace built at Heroult, Cal.

The electrodes, six in number, were spaced equidistant around the furnace, so that the current passing between them melted the charge, and the molten metal and slag were collected in the crucible, from which they were drawn as in ordinary blast-furnace work. The tap holes were arranged at different levels, so that the crucible might be partly or completely emptied, as desired.

The electrical equipment consisted of three oil-insulated, water-cooled transformers, having a rated capacity of 750 kilowatts each, 60 cycles, 2,200 volts primary, and a secondary range of 35 to 75 volts, constant output. The secondary current varied from 10,000 to 21,400 amperes.

The range in voltage was controlled by a dial switch in the primary circuit, giving steps of about 3 volts in the secondary. The efficiency was 98.6 per cent at 75 volts on the secondary, guaranteed for 25 per cent overload for two hours. The rise in temperature at full load, by test, was 35° C.

The generation of energy in the furnace was, therefore, controlled externally without movement of the electrodes, whose position was changed only to accommodate their wear in the crucible.

Many experimental runs were conducted in this furnace, and many changes and alterations were made in the construction and shape of the roof of the crucible, the manner of connecting the electrodes, the height and volume of the stack, the method of preheating the charge, the circulation of gases through the crucible and up through the shaft, and the like.

Although several hundred tons of splendid foundry iron was made in this furnace, much of the product was white iron, for the production of which this type of furnace seems best suited, as has also been ascertained at Trollhättan.

PRESENT TYPE OF FURNACE.

In the spring of 1911, owing to electrode troubles and to the irregular composition of the iron produced, the company decided to abandon the type of furnace that had been developed and to attempt to evolve a furnace in which iron could be produced that would better meet the requirements of the Pacific coast foundries. In this connection it may be stated that scrap iron is largely employed in that part of the country for making all grades of castings and that the sulphur, phosphorus, and combined-carbon content of many of the castings is so high as to impart very undesirable qualities to the finished product. For this reason the Pacific coast foundrymen have been in need of an iron high in graphitic carbon and silicon and low in other impurities. The Noble Electric Steel Co. states that it is able to produce such a grade of iron in the type of furnace now being used, and that the following is a fair average analysis of the pig iron that it is producing:

	Per cent.
Si	3.64
S	.00
P	.02
Combined carbon	.00
Total carbon	3.58

The details of the construction of this furnace have been furnished by the company, and are presented below, being taken chiefly from a recent article by Van Norden:[a]

THE PLANT.

The furnace is housed in a heavy steel-frame building, sheeted with corrugated galvanized iron. It is 120 feet long and 75 feet wide, and about 60 feet high. The interior is divided into two sections or bays by a line of columns. In the south section is the pouring floor, on which the pigs are cast. Throughout this bay is operated a 50-foot traveling crane. This is electrically operated and has one 20-ton and one 5-ton hoist. A lifting magnet, having a capacity of 2,000 pounds, is used in picking up the pigs. It will lift about 1,000 pounds of hot pigs.

The north bay contains the electric furnaces, their transformers, and the necessary controlling equipment.

THE FURNACE.

The crucible is contained in a steel shell 27 feet long, 13 feet wide, and 12 feet high. The crucible is rectangular, with a sloping bottom to facilitate the flow of the molten bath, when the furnace is tapped. The tap hole is in the center of the front of the crucible. The roof of the furnace is arched, and is penetrated by five stacks 24 inches in diameter and extending 15 feet above the roof of the crucible. In the four spaces between the stacks at the center of the dividing arches are inserted graphite electrodes. The electrode jackets and the arched roof are water cooled. The stacks are used only for charging, no reduction being attempted in them. A charging platform is built along the top of the stacks and supports a track which leads to the mixing platform. The loaded car is run along this track and the charge dumped directly into the stack. Except when receiving a charge, the stacks are closed with tight-fitting caps.

THE ELECTRODES.

Graphite electrodes are used. They are cylindrical in form, 12 inches in diameter, and 4 feet long. The upper end has a tapered male-threaded nipple, and the lower end has a corresponding socket with a female thread. As the electrode is fed into the charge a new one may be fastened to it, making a continuous feed.

An electrode in continuous operation lasts 30 days. Occasionally an electrode breaks in the furnace. Such breakage was formerly a serious matter and caused considerable delay and annoyance in the operation of the furnace, but this difficulty has now been practically obviated.

[a] Van Norden, R. W., Electric iron smelting at Heroult, Cal.: Jour. Elec. Power and Gas, Nov. 23, 1912.

THE TRANSFORMERS.

In the rear of the furnace, and as close thereto as is possible, are the three service transformers which supply three-phase current at 40 to 80 volts to the electrodes. These transformers are oil immersed and water cooled and have a capacity of 750 kilowatts each. The low-tension connections to the electrodes consists of eight pieces of flat copper bar, three-eighths of an inch thick and 5 inches wide, bolted together. On the 2,400-volt primary side there are brought out eight current taps for voltage regulation. These lead to an oil-immersed switch group, each unit of which is operated by a solenoid. This arrangement, with autotransformer compensator, gives 15 steps for voltage variation.

ELECTRIC CONTROL.

Electric control is through a switchboard, there being a panel for each furnace. As the current and power factor in each phase must be under observation at all times during operation, separate meters are installed in each phase. The requirements for one panel are 3 ammeters, 3 voltmeters, 3 wattmeters, 3 power-factor meters, and 3 recording wattmeters. The meters are mounted across the panel in rows of three each. Under the first four sets named are three hand-wheels to control the voltage variation, and under these three switches that control the entire load, and under the latter are the recording wattmeters.

For operating the voltage control and the main circuit breakers there is a 7½-kilowatt motor-generator set, comprising a 125-volt direct-current generator, directly connected to a 10-horsepower induction motor. This set has a small panel board mounting a circuit breaker, ammeter, voltmeter, and two single-pole knife switches. In the event that line voltage should fall, or if for any other cause the direct-current supply should become deranged, there is a storage-battery set, having a capacity of 7½ kilowatts, which may be instantly switched in, and thus prevent the furnaces from cutting out in the case of low voltage.

OPERATION.

The operation of the furnace is continuous. After a period of eight hours the hearth contains a full bath of molten metal, and therefore the metal is tapped three times each day. Charging is done at regular intervals, and the current is never shut off during operation. During the period of smelting the change in electrical conditions is interesting. At the beginning of the charge the power factor is almost unity. This gradually lowers as melting continues, until with a full bath of metal a power factor of 65 per cent is reached. If coke is used in place of charcoal, or if a mixture is employed, a dif-

ferent set of power-factor conditions exists. By studying these conditions it is possible to know the exact condition of the charge by looking at the meters. The load is, of course, a function of the voltage, and with half voltage the load will drop one-quarter.

In charging, the ore cars are run on the mixing floor, which is level with the charging floor. Charcoal and lime and quartz for flux are brought in on the lower floor in cars, hoisted on an electrically operated elevator to the mixing floor, and are dumped into their respective bins. The mixing is done in a car that is run on platform scales. The charge is placed in layers, the proportions depending on the tests made in the laboratory by the chemist. Following is a representative charge: Five hundred pounds of iron ore (magnetite); 135 to 150 pounds of charcoal; 3½ pounds of lime (well burned); 12½ pounds of quartz.

USE OF CHARCOAL AS A REDUCING AGENT.

Although, as explained later, the use of the electric furnace in the production of pig iron permits the manufacture of 3 tons of pig iron from 1 ton of coke or charcoal, the cost of a suitable reducing agent is still an important factor, especially in California. Charcoal is used at the plant of the Noble Electric Steel Co., and its economical manufacture in retorts so constructed as to save the distillation products has been almost as much of a problem as the production of the iron itself. Pit charcoal was used at first. A battery of beehive charcoal ovens, each holding 60 cords of wood, was then constructed. Later it was decided to use by-product retorts, and a vertical retort system was accordingly built. The retorts were arranged in two batteries of four each, each retort being a vertical cylinder 6 feet in diameter and 16 feet high. The volatile products were led from the bottom of the retorts to condensers. Altogether, wood distillation and the attendant recovery and working up of the by-products has not seemed to be a profitable enterprise on the Pacific coast. Soft woods are used for the most part, and these are generally lean in acetic acid and wood alcohol, although fairly rich in tarry products.

A splendid grade of charcoal was made at the plant of the Noble Electric Steel Co. in the retorts above mentioned, but the system proved to be rather cumbersome, and the time necessary for retorting was excessive. Moreover, owing to the manner in which the retorts were constructed, all the products resulting from distillation of the wood had to pass down through the hottest part of the retort. As a result the volatile products were broken up by heat, undesirable products were thus formed, and an inferior grade of by-products was produced. Consequently the company has decided to replace this system with what is known as the Yost system. The retort of the Yost system consists of a horizontal steel cylinder, about 5½ feet

in diameter and about 20 feet long, mounted in brickwork. One end of the retort is closed and the other is provided with doors. The wood is loaded on a car, and the car is run into the retort, which is then closed and sealed. The volatile products resulting from the heating of the wood are conducted to the condensers by means of a copper pipe; the liquid tar formed in like manner is drained from the bottom of the retort into a tank. Owing to its simplicity this type of retort is claimed to be much less expensive than the type formerly used, and because of the fact that the wood will not have to be handled as much as formerly the cost of producing charcoal is claimed to be much less.

USE OF CRUDE PETROLEUM AS A REDUCING AGENT.

As is well known, California is favored with a seemingly abundant supply of petroleum. Due to the scarcity of suitable reducing agents such as are at present used in metallurgical work attempts to use crude oil for this purpose have from time to time been made. It is quite possible that a successful process of this kind will be devised. The Bureau of Mines has been conducting investigations along this line and expects to publish the results in the near future. As soon as a commercially successful process of this nature has been devised, it is quite likely that the electric smelting of iron ores on the Pacific coast will receive an impetus, for the use of crude petroleum will not only permit the introduction of electric smelting in those districts where its use, except at prohibitive prices, is not now possible owing to the lack of suitable reducing agents, but should also cheapen the cost per ton of metal produced as compared with the cost when charcoal is used.

COMPARISON OF THE CALIFORNIA AND SWEDISH FURNACES AND PROCESSES.

As to the construction of the two types of furnaces, the difference is so apparent that no comment is necessary. As to the process, the California practice differs from the Swedish practice very decidedly in the following respects, namely: No attempt is made to reduce the ore in the stacks of the furnace; no artificial circulation of gases is employed; limestone calcined outside of the furnace is used. In other words, the California practice is just the reverse of the Swedish practice.

REDUCTION IN SHAFT.

As has previously been noted, the ideal condition in electric reduction furnace work would seem to be to have the charge in such a condition as to require only melting by the time it comes in proximity to the electrodes. The electric current would then be used to furnish only the additional heat necessary to bring about the melting

of the hot previously reduced oxide. Of course the heat for the reduction must come from some source, and its origin would be in the crucible and as a result of the heat developed during the melting of the charge, but the excess heat so developed can be transferred by the circulation of the gas to the charge in the shaft instead of being carried away by radiation and by circulating water.

CIRCULATION OF GASES.

As has also been mentioned on page 16, in the Swedish practice a part of the gases rising from the top of the shaft is returned to the crucible and made to pass through it and up through the charge in the shaft. This is done to cool the superheated brickwork of the roof of the crucible and to carry through the stack a volume of gas large enough and hot enough to bring about reduction in the stack.

As stated elsewhere, one of the most serious difficulties that had to be overcome in the development of the Swedish type of furnace was the destruction of the crucible resultant on local heating in the vicinity of the electrodes. Therefore, by returning to the crucible a part of the gases resulting from reduction, it is possible to prevent excessive local heating and at the same time to make use of the heat instead of wasting it. Although gas circulation has its objections, its advantages in the way of economy of operation would seem to outweigh its disadvantages. On the other hand, if reduction previous to smelting be considered the proper practice, this reduction will have to be made in the stack, and the present California practice will not find application except when warranted by the local conditions. As has been previously stated, the present demand in California is for a soft gray foundry iron. Although the furnace now used gives that product, such a construction and such a process are, of course, not absolutely necessary for the production of such an iron.

That the California type of furnace is better suited to the production of a soft gray foundry iron than is the Swedish type is probably due to the fact that reduction is performed solely by solid carbon, and the reduction of silica to silicon takes place at the same time as the reduction of iron oxide to metallic iron. The silicon is dissolved in the iron and by its presence causes the carbon to be precipitated as graphitic carbon, and there is thus obtained the soft gray iron desired. It is also quite probable that in a furnace having a rectangular crucible, such as has the California furnace, the temperature of the charge in the crucible as a whole is much higher than in a furnace of the Swedish type, and this higher temperature is more favorable to the reduction of silica to silicon. As before stated, the latter reaction is largely the controlling element in the production of soft gray foundry iron.

USE OF CALCINED LIMESTONE.

The use of calcined limestone has been repeatedly suggested, and theoretically seems desirable, but in operating a furnace of the Swedish type there are two objections to its use, namely, it increases the proportion of fine material in the charge and it makes the charge hang.

As to the former of these objections, the idea seems to be prevalent that there is no limit to the percentage of fine material that may be used in the electric reduction furnace. However, judging from observations of the smelting of black sands when the furnace consisted simply of a crucible and no shaft, some difficulty was experienced on account of the charge being made up entirely of fine material. In the Swedish practice the proportion of fine material that can be used in a charge was definitely determined in the work at Trollhättan, and it was found that a large proportion of fine material was detrimental to smooth running and good results; but Leffler, one of the engineers in charge of the work at Trollhättan, is of the opinion that calcined limestone may be advantageously added to the charge if no reduction is attempted in the shaft.

In view of the above facts, it would seem that the California practice as compared to the Swedish practice presents the following advantages: It permits the use of limestone calcined outside the furnace and it does not require the circulation of gases.

As to the circulation of gases and also as to reduction in the shaft, the California practice might perhaps prove more efficient if both were done. Although a very complete record of the working of the furnace at Trollhättan has been published, no such record of the operation of the furnace at Heroult is available, and so it is not at present possible to make a definite comparison of the two practices.

ELECTRIC FURNACE AS COMPARED TO THE BLAST FURNACE.

As has been mentioned in the section pertaining to the history and evolution of the electric pig-iron furnace, the electric furnace was not developed as a competitor of the blast furnace, but for the purpose of finding a furnace and a process that would be able to produce iron in those localities where blast-furnace practice was not feasible, or, as in Sweden, where the increasing cost of suitable fuel was becoming prohibitive to the existing practice of smelting in blast furnaces.

Broadly speaking, it may be said that the feasibility of smelting iron ores in an electric furnace depends upon the relative cost of either charcoal or coke and of electric power. As regards the latter, it must be cheap. As is well known, electric power at the present time can be developed more cheaply than it could when Capt. Stassano made

his experiments in 1898, and it will probably be produced still more cheaply in the future. However, the present purpose is not to speculate on this point but to state the facts as they are now understood. As has been previously mentioned, the average consumption of charcoal or coke in the electric furnace in the production of 1 ton of pig iron is about one-third of what it is in the blast furnace. Knesche[a] has pointed out that when one takes into consideration the consumption of electrodes, the saving is practically 0.7 ton of coke or charcoal per ton of iron produced.

HYPOTHETICAL CASE.

To make the matter more definite a hypothetical case is assumed, as follows: Coke is assumed to cost $6 a ton and a kilowatt-year to cost $16. From the report of the work done at Trollhättan it is learned that, on an average, 2,391 kilowatt-hours is required to produce 1 ton of pig iron. Hence, the cost for power per ton of pig iron produced would be $4.37, to which must be added the cost of 0.3 ton of coke and likewise the cost of 11.6 pounds of electrode, or an itemized cost as follows:

2,400 kilowatt-hours, at $16 per kilowatt-year	$4.37
0.3 ton of coke, at $6	1.80
11.6 pounds of electrode, at $0.06	.70
	6.87

As 1 ton of iron can be produced in a blast furnace from 1 ton of coke, the respective costs in the blast furnace and in the electric furnace would be $6 and $6.87. Hence, if considered on this basis alone, the electric furnace, with coke at $6 per ton and power at $16 per kilowatt-year, could not compete with the blast furnace. There are, however, other items to be taken into consideration, such as initial cost of plant, quality of iron produced, and especially the efficiency of the furnace, for, as has been pointed out by Ashcroft,[b] "the efficiency of many nonelectrical furnaces is barely 10 per cent of the theoretical, and very few will exceed 25 per cent, whereas the efficiency of electrical appliances sometimes reaches 75 per cent and is often 50 per cent."

Hence, with no greater difference between the respective costs than is shown above, a careful investigation of all items that enter into the total cost of production of a ton of pig iron, as well as the nature of the ore to be treated and the grade of product desired, might disclose that it would be more profitable to produce pig iron in the

[a] Knesche, J. A., Electric smelting of ore in the United States: Iron Trade Rev., Jan. 5, 1911, p. 65.

[b] Ashcroft, A. E., The influence of cheap electricity on electrolytic and electrothermal industry: Trans. Faraday Soc., vol. 4, 1908, pp. 134–142.

electric furnace than in the blast furnace. There are, however, few localities where coke costs only $6 a ton and where the cost of an electrical kilowatt-year is $16. In more localities the cost of coke is $12 to $14, and with coke at these figures the advantage is decidedly with the electric furnace, with power at $16 a kilowatt-year.

COST OF POWER.

In those electric-furnace iron plants that are operated at the present time only hydroelectric power is used. The cost of producing power for electric furnace work must, of course, vary with local conditions and hence depends on the initial cost per kilowatt of installation. In general there are few localities where the electric smelting of iron ores would be feasible with the electrical energy costing more than $20 to $30 per kilowatt-year.

USE OF ELECTRIC-FURNACE PIG IRON IN THE OPEN-HEARTH FURNACE.

The first metal produced at Trollhättan did not look much like pig iron and, moreover, as its analysis was different from that of the metal that had been used, doubt was expressed as to the possibility of using it in the production of steel. The first attempt to use the iron in the production of steel was made at Degerfors, where a basic furnace was used. The iron of the electric furnace was mixed with ordinary gray iron and a smaller quantity of scrap than usual. This mixture was made so as to insure a hot metal upon melting down. The results exceeded expectations, and on the third trial the charge was composed of only the electric-furnace metal and scrap. Much to the surprise of those operating the furnace, it was observed that the boiling which generally takes place immediately after the metal is melted was not so violent as usual. Tests were then made on the electric-furnace metal in an acid furnace. Although small quantities of the metal were at first used, it was soon ascertained that the charge could be composed of the ordinary proportions of 64 per cent pig iron and 36 per cent scrap.

Pig iron of normal composition was finally obtained as a product of the Trollhättan furnace, and when this iron was used for steel melting in the open hearth, together with the same proportion of scrap as had been used in the previous tests, a longer time was required to complete the refining of the metal, and it was necessary to add more ore than in the previous tests.

As a result of the tests at Degerfors the conclusion was reached that a metal could be produced in the electric reduction furnace that would be more suitable for steel making than ordinary pig iron.

In order to understand the reason for this conclusion, it may be well to note briefly some of the reactions that take place in the open hearth when treating pig iron of normal composition as compared to those which take place when treating pig steel.

In converting normal blast-furnace pig iron into steel in the open-hearth process, the pig iron must contain enough silicon to protect the iron from oxidation during the melting, and likewise enough silicon and manganese to free the iron after melting from the oxides and gases that were formed and taken up as a result of the action of the blast during the production of the pig iron in the blast furnace. The silicon and manganese that may be present in excess of this amount only prolong the time required to finish the heat.

During and immediately after the melting the metal boils violently, especially in an acid furnace if a low-silicon iron be used, even though it is not spongy. There is a second and less violent boiling when the metal is charged into the ladle and molds, especially in the case of hard steel. On the other hand, when electric-furnace "pig steel" is heated in the open-hearth furnace process, the second boil does not take place, and the regular boil commences immediately after the charge is melted and before the ore is added. Several theories have been advanced to account for such behavior of "pig steel" upon melting. One theory is that the metal is practically free from minute particles of oxides that are generally present in ordinary pig iron and that, as previously stated, are produced by the reoxidizing effect of the blast. Owing to the absence of the oxides, practically no reaction with the silicon and manganese present occurs at this time, and on this account the second boil in the open-hearth furnaces during and after the melting, when treating electric-furnace metal, does not occur, and it is possible to decarburize the charge at once by ore additions. The presence of silicon in such "pig steel" is therefore unnecessary during the first part of the open-hearth process.

As above stated, in the tests at Degerfors it was found that more time and more ore were required in the treatment of normal electric-furnace pig-iron than in the treatment of normal blast-furnace pig-iron. This is no doubt due to the fact that inasmuch as electric-furnace pig-iron of normal composition contains no oxides it is not necessary that silicon and manganese be present in the iron in order to take up oxides, and hence the removal of the silicon must be accomplished principally by means of ore, and, as pointed out by Odelberg,[a] only after this is accomplished can decarburation proper begin.

In conclusion it may be said that "pig steel" made in the electric furnace is better suited to the production of open-hearth steel than is the normal blast-furnace pig iron and that normal pig iron made in

[a] Odelberg, E., The behavior and qualities of the electric pig iron in the open-hearth process: Metall. Chem. Eng., vol. 9, 1911, p. 509.

the electric furnace is less suited to the production of open-hearth steel than is normal blast-furnace pig iron.

It must not be inferred from the above statements that it is not possible to produce normal pig iron in the electric furnace. As a matter of fact some hundreds of tons have been so produced in this country and in Sweden. Although the furnace at Trollhättan is so operated as to produce "pig steel," because this metal is better suited to steel making than is normal pig iron, the Noble Electric Steel Co., of California, on the other hand, so operates its furnaces as to produce a soft foundry iron which is particularly suited to the foundries of that section of the country. However, it has been found that a greater output may be obtained when making "pig steel" per unit of electrical energy expended than when foundry iron is produced.

AN ESTIMATE FOR THE ERECTION OF AN ELECTRIC IRON-SMELTING AND STEEL-REFINING PLANT.

An estimate of the equipment and cost of erection of an electric iron-smelting and steel-refining plant, with estimates of the cost of production of pig iron and steel in such a plant is here presented from data furnished by Electro-Metals, of London.

ASSUMPTIONS AND CONDITIONS.

The estimate is based on the following assumptions:

OUTPUT.

Annual production of pig iron, 50,000 tons; annual production of steel ingots, 50,000 tons.

SIZE AND NUMBER OF FURNACES.

Five electric shaft furnaces of 3,500 electric horsepower.

One tilting, open-hearth furnace of 50 tons capacity heated by gas from the shaft furnaces.

Three electric steel furnaces of 1,500 electric horsepower and of 10 tons capacity, one being kept as reserve.

ELECTRIC POWER.

For electric shaft furnaces: 18,000 electric horsepower (includes 500 electric horsepower for gas blowers, motors, etc.).

For electric steel furnaces: 3,000 electric horsepower.

The price of electric power is assumed to be $12 per electric horsepower-year. The electric power, 3-phase, of preferably 25 periods, is to be delivered at a high voltage from the water-power plant and to be transformed to a voltage of 50 to 100 volts close to the furnaces. For distribution to the electric motors a higher voltage of 500 to 1,000 volts should be used.

ORE.

To be supplied from local mines.
Iron content, about 60 per cent.
Cost, $0.50 per ton at the works.
Total quantity smelted per year, 80,000 tons.

FUEL.

For electric shaft furnaces, either coke or charcoal from local distilleries of wood alcohol.
Cost of fuel, $5 per ton at the works.
Total quantity consumed per year, 17,000 tons.

LIMESTONE.

From local quarries, lime rock, $CaCO_3$ content, 96 per cent.
Total quantity used per year, 8,000 tons.
Cost, $1.50 per ton, including calcining.

PROPOSED SYSTEM OF WORKING.

The pig iron produced in the electric shaft furnace is transferred either by ladle or, better, direct through launders, into a tilting open-hearth furnace, in which by addition of ore and lime the greater part of the carbon is removed and the temperature raised to about 1,500° C. The open-hearth furnace has a capacity of 50 tons, and also serves as a mixer, from which at regular intervals about 10 tons of partly refined steel is transferred to the electric steel furnaces. In these the metal is further heated to 1,600° to 1,700° C. and refined from any excess of sulphur not removed in the open-hearth furnace. The chief advantage of the electric steel furnace for the final treatment lies in the possibility of maintaining a reducing atmosphere in which reduction of the steel is more complete than in any other kind of steel furnace.

After being treated in the electric steel furnace for 1 to 2 hours the metal is recarburized to the desired carbon content, the ferro-alloys are added, and the metal is cast in the molds. The estimate provides also for fully equipped repair shops.

ITEMS ENTERING INTO ESTIMATED COST OF CONSTRUCTION.

In estimating the cost of construction of the plant the following items have to be taken into consideration:

Excavations for buildings and furnaces, grading and fencing, railway connection to works with crane for handling materials.

Light railway inside works with turntables, carriages, weigh-bridge, etc.

Water-supply plant with overhead storage tank and water pipes to gas scrubber, furnace water jackets and transformers, and pump with electric motor.

Buildings for furnaces and transformers.

Hoists for ore and lime, storage building for coke, crusher house and hoist.

General stores, offices, and laboratory.

Houses for officials and workmen.

Five electric shaft furnaces for 3,500 estimated horsepower, with foundations, iron constructions, brickwork, and lining.

Electrode holders with water jackets, lifting and regulating arrangements for the electrodes.

Gas blowers with electric motors, gas scrubbers, gas piping.

One tilting open-hearth furnace of 50 tons capacity with rockers and stands, hydraulic tilting cylinder, port ends, chills, gas and air connections and valves.

Working staging.

Iron chimney.

All silica, magnesite, fire, and red brick, silica and fire clay for brick setting, and magnesite.

Three electric steel furnaces, each of 10 tons capacity and 1,500 estimated horsepower, with foundations, iron construction, brickwork and lining, electrode holders with water jackets, automatic regulators, hydraulic tilting cylinders, working stagings, etc.

Two electric cranes of 20 tons carrying capacity.

Transformers for electric shaft and steel furnaces, with oil coolers, regulators, instruments and energy meters, lighting arresters, cables between transformers and furnaces.

Crushing plant with ore and coke crushers and electric motors.

Rails, wagons, weighing machines, tools, and sundries.

Transformers for small motors and for light, including wiring accessories.

Laboratory outfit and office fittings.

Superintendence during construction, drawings, and license.

Contingencies.

The total cost of erection of a plant of this description is estimated at $850,000.

COST OF PRODUCTION.

The estimated cost of producing pig iron and steel in a plant of this description is presented below:

Estimated cost of producing 50,000 tons of electric pig iron a year.

Item.	Weight		Cost.		
	Per ton of ore.	Total.	Per unit.	Total.	Per ton of pig iron.
Ore, tons	1.6	80,000	$1.50	$120,000	$2.39
Coke, tons	.33	17,000	5.00	85,000	1.68
Lime, tons	.16	8,000	1.50	12,000	.20
Electrodes, pounds	15	335	75.00	25,125	.52
Power (18,000 horsepower year)			7.50	135,000	2.72
Staff and labor:					
3 foremen, 2 electricians, and 105 workmen				75,000	1.50
Maintenance and repairs				15,625	.31
Laboratory				3,125	.06
Office expenses				3,125	.06
Depreciation				28,125	.56
Royalty				37,500	.75
Contingencies				25,000	.50
Total				564,625	11.25

Estimated cost of producing 50,000 tons of electric steel ingots a year.

Item.	Weight.		Cost.		
	Per ton of steel.	Total.	Per unit.	Total.	Per ton of steel.
Pig iron, tons	1,000	50,000	$11.29	$564,625	$11.29
Iron ore, tons	0.112	5,600	1.50	8,400	.16
Burnt lime, tons	.1	5,000	3.00	15,000	.29
Ferro-alloys, tons				25,000	.50
Electrodes, pounds	20	450	75.00	33,750	.68
Power (3,000 horsepower year)			7.50	22,500	.45
Staff and labor:					
3 foremen, 2 electricians, 50 workmen				37,500	.75
Maintenance and repairs				37,500	.75
Ladles, stoppers, molds, and other tools				50,000	1.00
Laboratory and office expenses				6,250	.12
Depreciation				28,125	.56
Royalty				18,750	.37
Contingencies				31,250	.62
Total				878,650	17.54

THE ELECTRIC FURNACE IN STEEL MANUFACTURE.

By ROBERT M. KEENEY.

INTRODUCTION.

The purpose of the second part of this report is to present a brief historical review of the development of the electric-furnace manufacture of steel up to the present time; to describe in detail the types of electric furnaces in commercial operation for the manufacture of steel and, in general, types which have not yet attained wide use; to give a description of the practice of European and American electric-furnace steel plants that were recently visited by the writer; to compare in a general way the different types of furnaces and the more established methods of steel manufacture with the electric-furnace process; and to discuss various problems of the electric-furnace manufacture of steel and the possible future developments of the process.

HISTORY OF THE DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

EARLY DEVELOPMENT OF THE ELECTRIC FURNACE.

The process of evolution through which the electric furnace passed before reaching its most highly developed use, which is the manufacture of steel, really began with the furnace of Siemens, the prototype of the modern arc furnace. In 1878 Siemens patented the furnace and operated it in a very small laboratory way. This electric furnace consisted of a crucible of refractory material with a movable vertical electrode, the other electrode being an iron bar passing through the bottom of the crucible. One electrode was connected to the positive pole and the other to the negative pole of a direct-current circuit. The crucible was surrounded by a heat insulator, and the bottom metallic electrode was water cooled.

In 1885 Ferranti devised a furnace the principles of operation of which were the fundamental basis of the modern induction furnace. He used the induced current of a magnetic circuit to heat and melt metals which were arranged like the secondary of a short-circuited transformer.

Although the electric arc and induced currents were applied to furnace heating by these two pioneers, no attempt was made to operate a commercial electric furnace for the manufacture of steel until 1898. In the meantime, in 1886, the Cowles brothers had devised their arc furnace for the production of aluminum alloys. In 1887 Colby patented an induction furnace for melting metals that was quite similar to the modern Kjellin furnace. In 1892 Willson designed a furnace for the manufacture of calcium carbide that was simply a Siemens crucible reproduced on a commercial scale. In 1893 the radiated heat of the electric arc from two horizontal electrodes was used by Moissan during the course of his extensive experiments on carbides and oxides of metals. From 1893 to 1898 the electric arc furnace was developed rapidly in the manufacture of calcium carbide, although it was still essentially the old Siemens crucible with an upper electrode of carbon and a carbon block in place of the lower metallic electrode.

In 1898, owing to overproduction of calcium carbide and the substantiation of the Bullier patents, many works were compelled to stop the manufacture of carbide, and either turn to the manufacture of other products or shut down, which meant the idleness of many hydroelectric plants capable of developing thousands of horsepower. The experiments of Moissan had shown the possibility of making ferro-alloys in the electric furnace. Ferro-alloys were made in the old carbide furnaces, and with the introduction of these electric-furnace alloys, which were of higher percentage and of greater purity than the blast-furnace product, a steady demand for them arose. As a result, by 1900 a new industry was in operation. In many plants both calcium carbide and ferro-alloys were made, as is the case as present.

EARLY DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

DEVELOPMENT OF THE HÉROULT STEEL FURNACE.

As the manufacture of ferro-alloys developed and the necessity of producing alloys of low carbon content became apparent the design of the electric furnace was gradually altered so as to make the operation possible without a conducting carbon lining. It was only a step from the Héroult ferrochrome furnace lined with chromite, with a bottom carbon electrode, over which metal was permitted to freeze to prevent carburization, to the nonconducting hearth of the modern Héroult steel furnace, shown in figure 18, as it had been found that this freezing of metal hindered the regular operation of the furnace. The fundamental principles of the modern Héroult steel furnace were patented in 1900.

EXPERIMENTS OF STASSANO.

In the meantime Stassano,[a] of Italy, through an interest in the carbide industry, had attempted to produce steel directly from iron ore in the electric furnace, with charcoal as a reducing agent. The experiments were begun in 1898 and were first conducted in a modified blast furnace, in which electrodes were substituted for tuyères. This furnace being unsuccessful, a new furnace similar to the modern Stassano steel furnace, shown in figures 19 and 20, was built, having horizontal electrodes and arcs heating the charge by radiation, as in

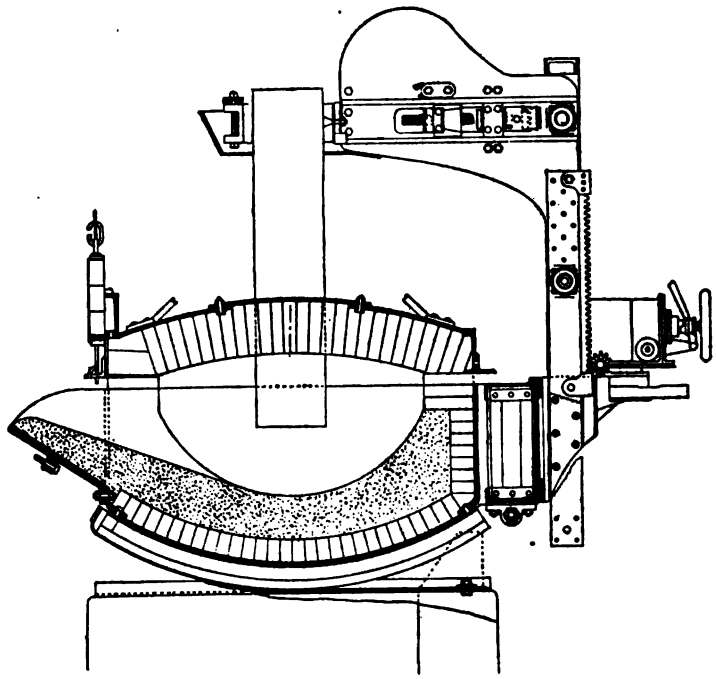

FIGURE 18.—Sectional elevation of 2.5-ton, single-phase Héroult steel furnace, La Praz, France.

the Moissan furnace. However, it was not well suited to reduction of the ores that were first used.

In these experiments a high-grade hematite ore of low phosphorus and sulphur content was used. The ore was first crushed and then briquetted with charcoal, lime, and tar. The steels produced contained about 0.10 per cent carbon, less than 0.024 per cent phosphorus, and not over 0.073 per cent sulphur. The power consumption aver-

[a] Stassano, E., Treatment of iron and steel in the electric furnace: Electrochem. and Metall. Ind., vol. 6, 1908, p. 315; Catani, R., The application of electricity in the metallurgical industry of Italy: Jour. Iron and Steel Inst., vol. 84, No. 2, 1911, p. 215; Metall. Chem. Eng., vol. 9, 1911, p. 642.

aged 4,205 kilowatt-hours per metric ton.[a] It will be seen later that this item of power consumption is over five times that of making steel from cold scrap in the electric furnace, a difficult handicap for

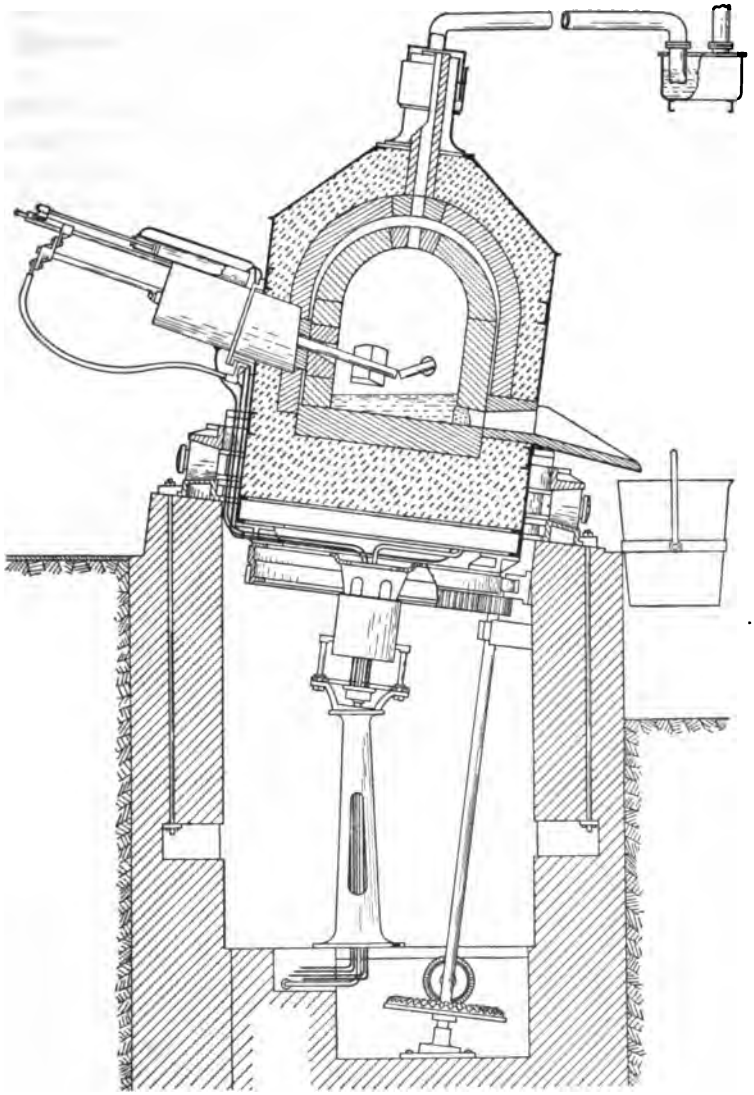

FIGURE 19.—Elevation of 1-ton, three-phase Stassano steel furnace, Bonn, Germany.

any direct electric-furnace steel process to overcome, except under the most favorable circumstances. Later experiments of others have reduced this item considerably.

[a] Throughout this report the metric ton, 2,204 pounds, is used in reference to European practice, and the long ton, 2,240 pounds, for American practice.

Stassano was the first to demonstrate that malleable iron or a fluid steel can be produced directly from a pure ore in the electric furnace. The power consumption was so high and the cost of briquetting so great as to prevent the commercial application of this process. Later, in 1908, Stassano showed by experiments the possibility of obtaining good steel from ore containing impurities, but the cost was still prohibitive.

From these experiments on iron ore, Stassano turned to the development of his furnace for the manufacture of steel from scrap iron and steel, and he had a furnace operating for this purpose at the Italian Government gun foundry in Turin by the time of the visit of the Canadian Commission in 1904.

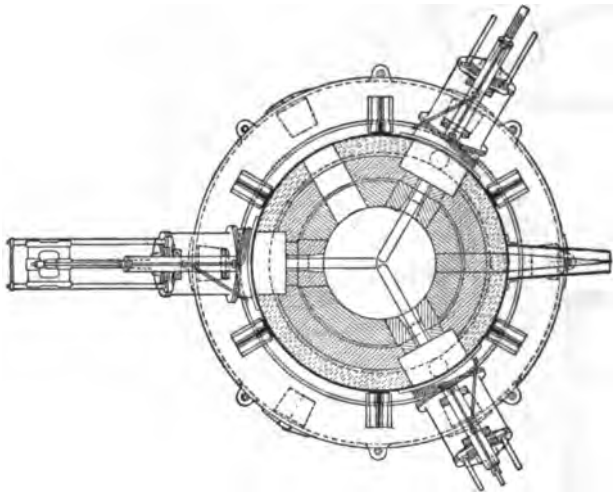

FIGURE 20.—Plan of 1-ton, three-phase Stassano steel furnace, Bonn, Germany.

THE KJELLIN FURNACE.

The application of the induction principle to the manufacture of steel was begun not because of an overdevelopment of any electric-furnace industry, but because of actual need of such a furnace in steel manufacture. In 1899 Kjellin, a Swedish electrical engineer, was requested by the works manager at Gysinge, Sweden, to construct an electric furnace. The furnace was to be lined for the production of the highest grade of tool steel from Dannemora iron manufactured at Gysinge. Kjellin did not approve of the arc electric furnace used in carbide manufacture, because he thought the contact of the electrodes with the charge would contaminate the product. A furnace was designed in which the molten steel lay in an annular ring that was used as the secondary coil of a step-down transformer. Kjellin conceived the construction independently and

did not know until later, when he applied for patents, that the same principle had previously been used for this purpose by Ferranti and Colby. To Kjellin, however, belongs the credit of constructing the first induction steel furnace that was a commercial success. Construction was started in May, 1899, and completed in February, 1900. On March 18, 1900, the first steel ingot was cast. The small 58-kilowatt furnace of 100 kilograms (220 pounds) capacity produced about 600 kilograms (1,320 pounds) of ingots per 24 hours, and was successful from the start. A larger furnace taking 165 kilowatts and producing 4 tons per 24 hours was started in May, 1902. In fundamental design these furnaces were practically the same as the modern Kjellin electric steel furnace (fig. 21).

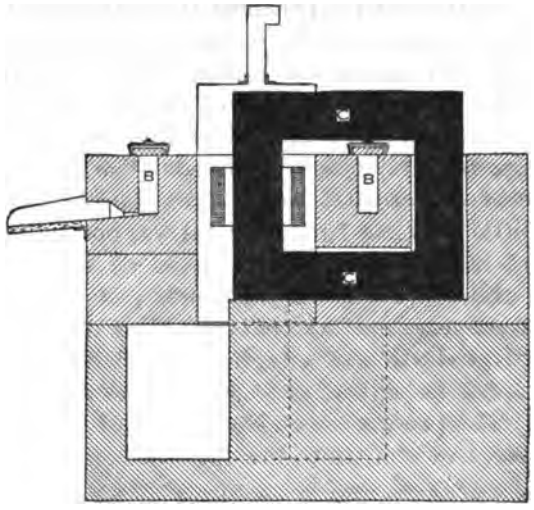

FIG. 21.—Elevation of 1.5-ton, single-phase Kjellin steel furnace, Gysinge, Sweden. C, laminated iron core of primary circuit; B, annular crucible.

INVESTIGATIONS OF THE CANADIAN COMMISSION OF 1904.

In 1904 the commission[a] appointed by the Canadian Government to investigate electric-furnace pig iron and steel processes in Europe found four electric furnaces being used for the production of steel at the following places: A Kjellin furnace at Gysinge, Sweden; a Héroult furnace at Kortfors, Sweden; a Héroult furnace at La Praz, France; and a Stassano furnace at Turin, Italy.[a] Such was the nucleus from which the modern manufacture of steel in the electric furnace has developed.

EXPERIMENTS WITH KJELLIN FURNACE, GYSINGE, SWEDEN.

At Gysinge three experiments were conducted in a 165-kilowatt Kjellin furnace of the type shown in figure 21. The first charge was arranged to give a high-carbon steel containing about 1 per cent of carbon. The charge consisted of high-grade Dannemora pig iron, bar scrap Walloon iron, and some scrap steel left in the furnace from previous runs. All of the materials had a low phosphorus

[a] Haanel, E., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe: Mines Branch, Department of the Interior, Canada, 1904, p. 1.

and sulphur content. After 6 hours' running about 1 ton of very high-grade steel containing 1.082 per cent carbon was poured. There was very little slag removed in the process. The power consumption was 857 kilowatt-hours per ton.

The object of the second experiment was to make a medium carbon steel containing about 0.5 per cent carbon. The same grade of material was charged as in the first run but in different proportions. After 6½ hours' running about 1 ton of very pure steel, containing 0.417 per cent carbon, was poured. A little more slag was removed than in the first run. The power consumption was 1,100 kilowatt-hours per ton.

In the third experiment the object was to make a low-carbon steel containing less than 0.2 per cent carbon. The charge consisted of bar iron with a low carbon content. Three attempts were made to get a clean pour, but owing to the power on the furnace being insufficient to give a temperature high enough for this grade of steel no precise results were obtained. The steel poured contained 0.098 per cent carbon and had a low phosphorus and sulphur content. A little less than 1 ton of metal was obtained with a power consumption of 1,430 kilowatt-hours per ton.

Kjellin estimated the power cost at this plant as $15.50 per kilowatt-year, or 0.18 cent per kilowatt-hour. On this basis the cost of production of a high-carbon steel of the first class was estimated at $37.48 per ton, or $5.82 per ton exclusive of the cost of material.

The process as conducted in 1904 was, as regards its metallurgical features, similar to the old crucible process, the desired variation in the finished steel being produced by altering the relative proportions of pig iron and scrap iron and little or no purification being attempted. The commission found the Kjellin furnace well adapted for the production of the highest class of steel from pure materials. It seemed to be limited in use to pure materials, and gave better results with high-carbon steel than with mild steel. There was a marked increase in power consumption when the percentage of carbon in the steel was decreased.

EXPERIMENTS WITH THE HÉROULT FURNACE, LA PRAZ, FRANCE.

At La Praz, France, two experiments were made on the production of both high and low carbon steel in a 320-kilowatt Héroult single-phase furnace with a capacity of 2½ to 3 tons (fig. 18), described on page 74. In the first run the materials consisted of scrap iron and steel containing 0.11 per cent carbon, 0.055 per cent sulphur, and 0.22 per cent phosphorus, and a slag of iron ore and lime for dephosphorizing. When completely melted the slag was poured off and two successive slags of lime, sand, and fluorspar were added to remove the sulphur. The charge was poured after a run of 4½ hours. About 1¼ tons of excellent steel was obtained, containing 0.079 per cent

carbon, 0.009 per cent phosphorus, and 0.022 per cent sulphur. This steel was for transformers. The yield was 84 per cent of the total charge of metal. The power consumption was 1,555 kilowatt-hours per ton.

The object of the second experiment was to produce a high-carbon steel. The charge was about the same as in the first, but after complete purification at the end of 5 hours and 20 minutes the bath was recarburized by the addition of carburite, a mixture of pure carbon and iron. Eight hours after starting about 2¼ tons of steel was poured. This yield was about 90 per cent of the charge of metal. The steel contained 1.016 per cent carbon, 0.02 per cent sulphur, and 0.009 per cent phosphorus. The power consumption was 2,840 kilowatt-hours per ton.

The electrode consumption was about 18 kg. (39.6 pounds) per ton. There was a 50 per cent loss in electrodes due to unconsumed ends. These, however, were mixed with an equal quantity of fresh coke to make new electrodes. Harbord[a] estimated the cost of converting scrap into steel at La Praz at $15.40 per ton of product exclusive of materials.

The operation at La Praz was decidedly different from that at Gysinge; at Gysinge high-grade steel was produced from pure materials, while at La Praz a high-class steel was made from low-grade scrap. The high power consumption at La Praz was due to the necessarily prolonged use of dephosphorizing slags. When scrap iron nearly free from carbon was used, the first product obtained was soft steel; to obtain high-carbon steel suitable additions of carbon had to be made. For this reason the consumption of energy was greater with high-carbon than with low-carbon products. The process at La Praz was the reverse of the one at Gysinge, where a high-carbon material was available at the start. The methods depend on the materials available. Although fine raw material had to be used at Gysinge, any sort of scrap could be used in the Héroult furnace because of the high temperature and the easy manipulation of slags. The commission found the Héroult furnace admirably suited to the manufacture of high-grade tool steel from impure materials.

CONCLUSIONS OF THE COMMISSION.

The conclusions reached in 1904 by Harbord,[b] metallurgist of the commission, in respect to the electric production of steel, were as follows:

1. Steel equal in all respects to the best Sheffield crucible steel can be produced either by the Kjellin or Héroult processes at a cost considerably less than the cost of producing high-class crucible steel.

[a] Harbord, F. W., Report of the commission appointed to investigate the different electrothermic processes for the smelting of iron ores and the making of steel in operation in Europe; Mines Branch, Department of the Interior, Canada, 1904, p. 50.

[b] Harbord, F. W., loc. cit., p. 115.

2. Structural steel to compete with Siemens or Bessemer steel can not be economically produced in electric furnaces, and such furnaces can be used commercially only for the production of exceptionally high-class steel for special purposes.

LATER DEVELOPMENT OF THE ELECTRIC STEEL FURNACE.

THE GIROD ARC FURNACE.

In connection with the development of the Girod ferro-alloys works at Ugine, France, Girod had devised an arc furnace with a noncarbon-conducting hearth that was used for the manufacture of low-carbon alloys. As the ferro-alloy industry became well established, Girod turned his attention to a steel furnace, and in 1905 constructed a furnace based upon the principles of the earlier ferro-alloy furnace. The furnace (fig. 22) has a conducting hearth consisting of magnesite, in which iron poles are embedded to serve as conductors. The poles are water-cooled at the lower ends. There may be one or more upper carbon electrodes. Thus the chief feature of the Girod furnace may be traced back to the lower water-cooled iron electrode of Siemens. Girod was the first to apply the noncarbon conducting hearth to the steel furnace.

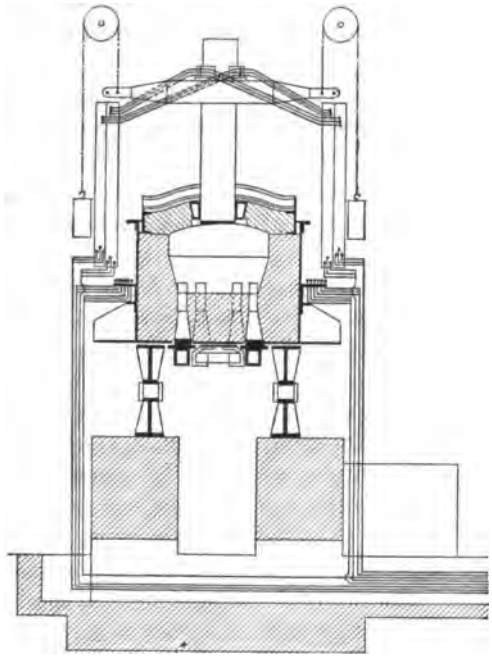

FIGURE 22.—Elevation of 2.5-ton, single-phase Girod steel furnace showing arrangement of conductors.

OTHER ARC FURNACES.

After the introduction of the noncarbon conducting hearth in 1905 other furnaces similar to the Girod were developed. The Keller furnace (fig. 23) is similar to the Girod furnace, except that there are numerous iron rods embedded in the hearth of magnesite about 1¼ inches apart, instead of about six poles, as in the Girod furnace. The whole mass of magnesite and iron acts as a conductor when hot.

Grönwall constructed a furnace (figs. 24, 25) having a conducting hearth composed of a mixture of dolomite and tar, and two upper car-

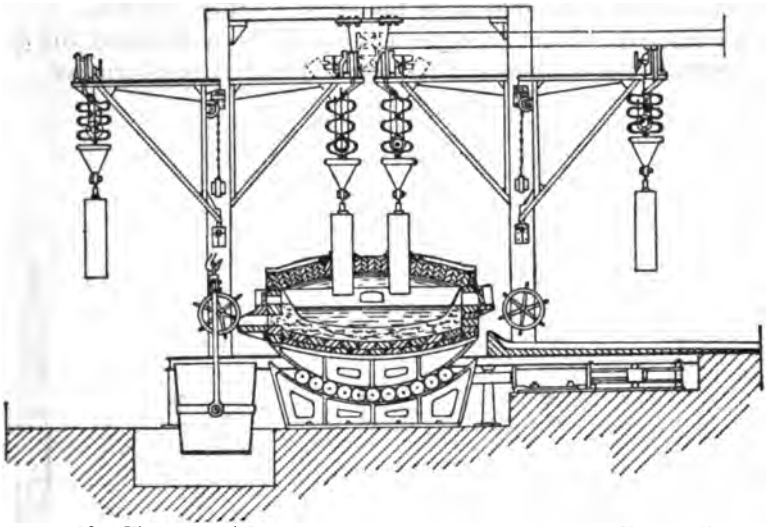

FIGURE 23.—Elevation of 8-ton, three-phase Keller steel furnace, Unieux, France.

bon electrodes which were each connected to a phase of a two-phase system, the hearth forming the neutral point of the system.

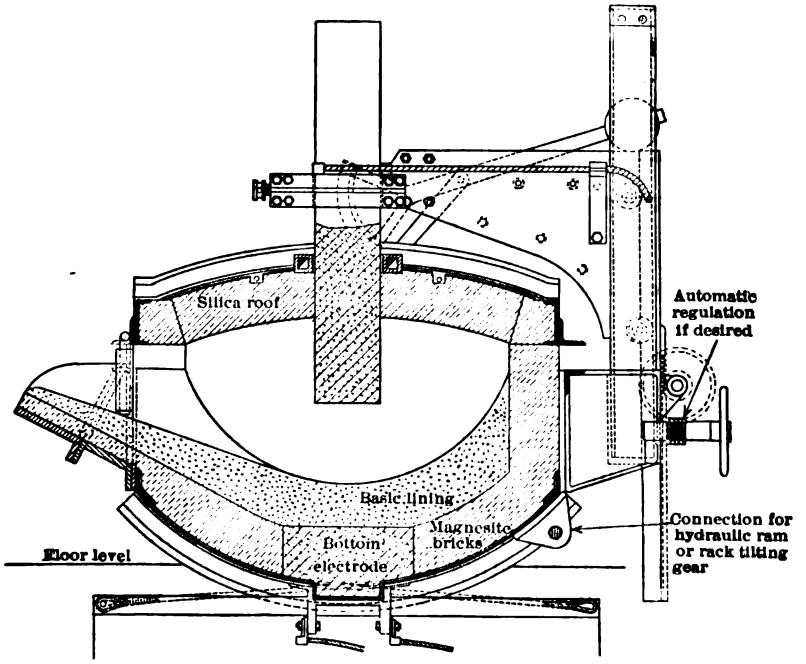

FIGURE 24.—Elevation of 2.5-ton, two-phase Grönwall steel furnace, Sheffield, England.

Nathusius has built a furnace (fig. 26) combining the principles of the arc and resistance furnaces, as the hearth contains three poles

of iron, each connected to a phase of a three-phase system, while there are also three upper carbon electrodes, each connected to a phase. This is the latest development of the arc-resistance furnace.

Other furnaces of the arc type which have been designed and built since 1904 are the Anderson, Chaplet, and Soderburg furnaces.

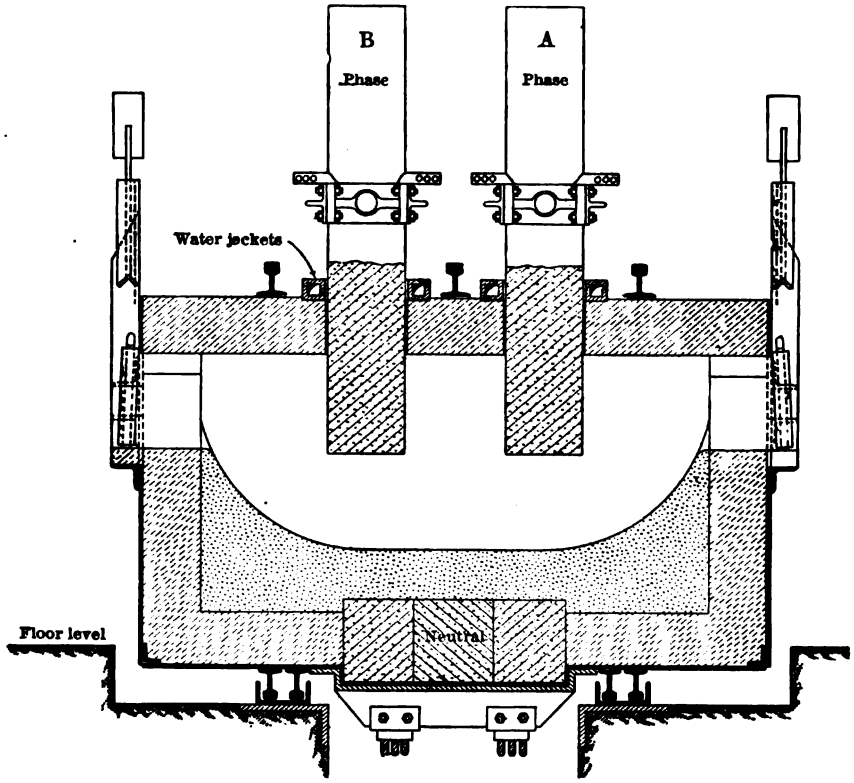

FIGURE 25.—Longitudinal elevation of 2.5-ton, two-phase Grönwall steel furnace, Sheffield, England.

THE RÖCHLING-RODENHAUSER FURNACE.

The Röchling-Rodenhauser electric steel furnace (fig. 27) was developed from the original Kjellin design in 1906 to meet the requirements for refining molten basic Bessemer steel. It combines two methods of electric heating—the ordinary heating by electric induction currents and heating by induction currents that are introduced into the molten bath through metal plates. It is the most widely adopted induction furnace of the present time, and the design is practically the same as the original. The Röchling-Rodenhauser

furnace was the first induction furnace that could be heated with a current of ordinary frequency without having too low a power factor, as had the old design.

OTHER INDUCTION FURNACES.

Other induction furnaces are the Frick, the Hiorth, and the Paragon furnace of Harden. Several furnaces of the first two types have been built, but their adoption has not been widespread. Greene[a] has devised a combination induction converter which has not passed beyond the experimental stage.

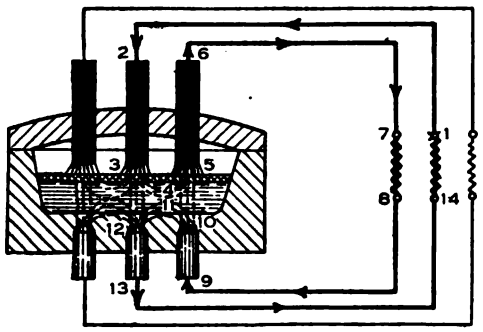

FIGURE 26.—Elevation of Nathusius three-phase steel furnace showing principle of operation.

GENERAL LINES OF DEVELOPMENT OF ELECTRIC STEEL FURNACES.

With the natural increase in size of units constructed there has been a marked tendency to use three-phase electric current in the

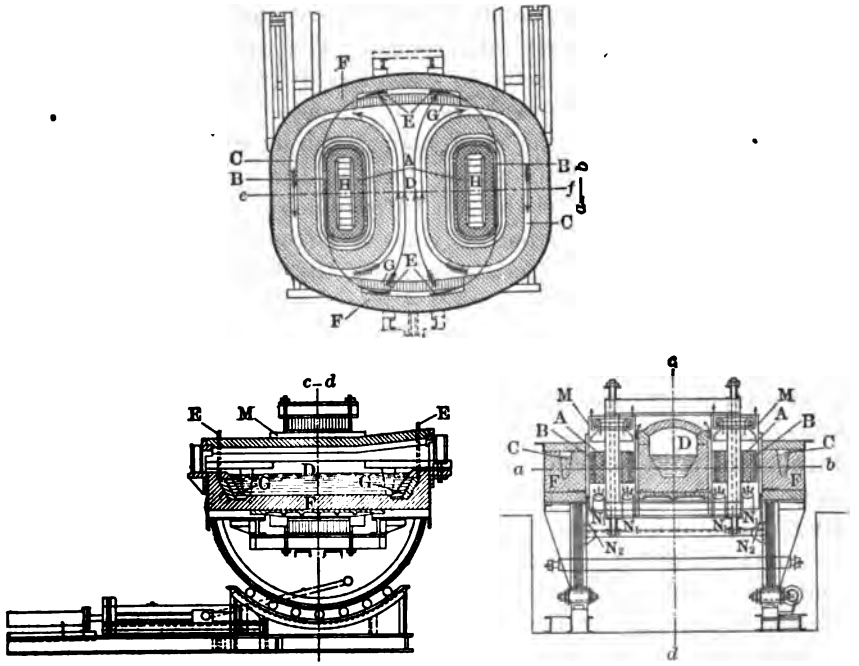

FIGURE 27.—Plan and elevation of 2-ton, single-phase, Röchling-Rodenhauser steel furnace, Volklingen, Germany. For explanation of lettering see p. 86.

larger furnaces of all types, except the induction furnace. Many of the recent furnaces have been designed especially with this in view

[a] Greene, A. E., Electric steel processes as competitors of the Bessemer and open-hearth; Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 233.

to save the expense of installing motor generator sets for transformation from three or two phase current to single-phase current. At present new generating units are invariably three-phase if the energy is to be transmitted a considerable distance. It is probable that a more even distribution of heat is obtained in a large furnace by the use of three-phase current. In induction furnaces it has been proved by experience that for units above 3 tons three-phase current does not give as good satisfaction as single-phase current, owing to the complicated nature of the electrical transforming part of the furnace.

Electrode consumption has been reduced considerably as a result of the production abroad of amorphous carbon electrodes in sizes up to 24 inches (61 cm.) in diameter, that screw into each other so that the butts are not wasted. Graphite electrodes of this design have been made in this country, but up to a recent date no domestic amorphous carbon electrodes threaded for continuous feeding were made. The strength and density of the carbon electrode are considerably greater than they were several years ago. The carbon electrode is used almost entirely by foreign electric-steel manufacturers, because of its greater cheapness.

Generally speaking electric steel furnaces were first operated by men not familiar with the steel industry, so that linings and roofs were not always properly constructed. The rather high cost item that resulted has been reduced considerably with the adoption of the electric furnace by established steel firms.

Under the hand of the steel manufacturer, the electric furnace seems to be gradually taking the form of the open-hearth furnace. The hearth proper of the Héroult furnace is like an open hearth, but the furnace is charged from the sides and front and is generally set back a short distance from the edge of the working floor. In the open-hearth furnace the charging door is on one side and the pouring spout on the opposite side, the furnace being level with the edge of the working floor. The electrode supports of the Héroult furnace as at present arranged prevent the use of a back charging door. The Girod furnaces at Ugine, however, are similar to the open-hearth furnace in these respects, as the electrodes are supported from the sides. A three-phase Héroult furnace now being constructed in England is to have the three electrodes suspended from the sides as in the Girod furnace, with a back charging door opposite the spout and close to the edge of the working floor. The Kjellin furnace is simply an annular crucible. The Röchling-Rodenhauser furnace has a rather large hearth in the center, and a charging door opposite the pouring spout.

The wider use of the electric process, together with the improvements mentioned, has resulted in a considerable increase in heating efficiency over the results obtained in 1904.

For a few years the induction furnace was developed as rapidly as the arc furnace, but during the past three years its use has not increased as rapidly. The reason for this will be discussed later.

PRESENT STATUS OF THE ELECTRIC STEEL INDUSTRY.

In 1904 there were 4 electric furnaces being used in Europe for the manufacture of steel, whereas to-day there are 114 electric furnaces producing steel in Europe and the United States and 30 others are in course of construction. As in other electrothermic processes, development has not been so rapid in the United States as in Europe. Only 14 furnaces are in this country. The average capacity per charge of the furnaces already built is 3.7 tons, whereas that of the furnaces under construction is 4.5 tons, an increase of 21.6 per cent. The total charge capacity of the furnaces now installed is about 250 tons per charge, and the total charge capacity of the furnaces under construction will be 170 tons per charge. The arc furnaces vary in capacity from 1 to 15 tons and require from 200 to 1,500 kw. for operation. A Héroult furnace of 25-ton capacity, requiring 3,000 kw., is nearly completed at Bruckhausen, Germany. The induction furnaces vary in capacity from 1 to 10 tons and require from 165 to 600 kw. for operation.

Of the 114 furnaces in operation 84 are arc furnaces and 30 are induction furnaces; of the 30 under construction, 26 are arc furnaces and 4 induction furnaces.

The following table gives the annual production of steel in electric furnaces, by countries, for the years 1908 to the first half of 1912.

TABLE 1.—*Yearly production of electric-furnace steel.*

Country.	1908			1909			1910			1911			1912, first half.
	Production.	Number of furnaces.	Change in production.	Production.	Number of furnaces.	Change in production.	Production.	Number of furnaces.	Change in production.	Production.	Number of furnaces.	Change in production.	
	Tons.		*Per cent.*	*Tons.*		*Per cent.*	*Tons.*		*Per cent.*	*Tons.*		*Per cent.*	*Tons.*
Germany and Luxemburg	19,536	8	.....	17,773	8	— 9.0	36,188	13	+104.1	60,654	15	+ 67.8	.....
United States	55	1	.....	13,762	4	.....	52,141	7	+228.0	29,105	9	— 44.2	6,882
Austria-Hungary	4,333	.....	.....	9,048	.....	+109.2	20,028	.....	+120.8	22,867	10	+ 14.0	.....
France	2,686	7	.....	6,515	12	+127.6	13,445	21	+106.2	13,850	21	+ 3.0	7,920
Sweden	.....	.....	.....	591	11	.....	431	12	— 27.0	2,034	13	+372.0	.....
Norway	.....	.....	.....	.....	.....	.....	.....	1	.....	.....	1	.....	.....
England	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	12	.....	.....
Italy	.....	.....	.....	.....	5	.....	.....	2	.....	.....	4	.....	.....
Switzerland	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	.....	.....
Belgium	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	2	.....	.....
Russia	.....	.....	.....	.....	.....	.....	.....	.....	.....	.....	1	.....	.....
Total	26,610	16	.....	47,039	40	+ 78.2	120,116	56	+155.4	126,476	90	5.22	.....

From Table 1 it may be seen that for a new process in so conservative an industry as iron and steel manufacture, progress has been very rapid since 1908. Germany leads all countries in the steady growth of the process and the total tonnage produced. Although in Germany the production of electric-furnace steel increased 67.8 per cent in 1911, in the United States it decreased 44.2 per cent. The decrease in this country was probably due to the conservatism of American steel makers, which has prevented the wide adoption of the process before experimental results have conclusively proved its merits. From present indications there will be a considerable increase in the production of electric-furnace steel in this country in the near future, although a very small tonnage, 6,882 tons, was reported by the American Iron and Steel Institute as the output for the first half of 1912.[a] Of the total production of electric-furnace steel in the United States in 1911, 27,227 tons were ingots and 1,878 tons castings. Of the total tonnage of electric-furnace steel made here in 1911, 6,700 tons were alloy steel and 462 tons were rolled into rails. The large production of steel in the United States and Germany in proportion to the number of furnaces operating is due to the use of molten Bessemer and open-hearth steel instead of cold scrap. The use of the latter almost entirely accounts for the comparatively small tonnage produced by France in proportion to the number of furnaces in operation. No figures were obtained for England, but it is probable that at least 10,000 tons of electric-furnace steel is manufactured in England. It is estimated that about 12 furnaces operate there, several of which receive hot-metal charges. Italy, Norway, Switzerland, Belgium, and Russia produce small tonnages also. The slight increase in the total electric-furnace steel production for 1911 over that produced in 1910 was caused by the big decrease in production in the United States.

In the first years of its development the electric process was considered as a competitor of the crucible process only for making high-class steel from scrap iron and scrap steel; but with the successful operation of larger furnaces the electric process is likely to become an important adjunct to the Bessemer and open-hearth processes as a means of superrefining the molten products that they yield. The electric process, however, does not appear to be destined to supersede either of these methods, as greater efficiency and economy are obtained by a combination of any two of the three processes as a duplex process. The success of recent experiments has obtained for the electric process a definite place as a superrefining method. In time preliminary refining will probably be done mainly in the Bessemer converter, the process being finished in the electric furnace or the open hearth. In Europe the electric-furnace process for making steel

[a] Iron Age, The world's output of electric steel; vol. 91, 1913, p. 304.

of the highest grades is rapidly superseding the old crucible method, because of its greater economy of operation and the possibility of using materials of lower grade.

ELECTRIC STEEL FURNACES.

In general electric furnaces in commercial use for the manufacture of steel may be divided into two groups—arc furnaces and induction furnaces. In the arc furnace the heating is caused by the arc, which may be between the electrode and the bath, or between two or more electrodes so arranged as to heat the metal by radiation only. In the induction furnace the heat is supplied by a current induced in the bath. The operation is similar to that of a step-down transformer, having a large number of primary turns and a single secondary turn, which is formed by the steel in the furnace.

ARC FURNACES.

THE HÉROULT FURNACE.

The Héroult electric steel furnace heads the list of electric furnaces in use in the iron and steel industry with 31 furnaces already built and 20 others in course of construction. The wide use of the Héroult furnace is due chiefly to its efficiency, simplicity of construction, and adaptability to many different uses. Also it was the pioneer electric steel furnace.

SINGLE-PHASE HÉROULT FURNACE.

The design of the 2½-ton, single-phase Héroult furnace has changed little from that (fig. 18) of the first Héroult furnace, which has been in operation at La Praz, France, continuously since 1900. The furnace consists of a shallow hearth of dolomite, similar to the open hearth, incased in a steel shell, and covered with a roof of silica brick. The 2½-ton Heroult furnace at Braintree, England, has the pouring spout in the center of one side, and two doors, one on each end of the furnace. There are two 14-inch carbon electrodes projecting into the furnace through the roof and held in place by a steel framework extending from the rear over the roof. The electric current arcs between each of the electrodes, which are connected in series, and the bath, thus passing through the bath. The Braintree furnace is operated with 300-kw. 25-cycle alternating current at 100 volts. The furnace is set on curved steel bars, the whole being on a concrete foundation, and is tilted by rotating a screw, operated by a 5-kw. electric motor. Springs are used on the bottom to keep the furnace from creeping.

The exterior of the 2½-ton furnace is 5 feet 6 inches wide, 7 feet 6 inches long, 4 feet 9 inches high to the top of the roof, and 9 feet

3 inches high to the top of the electrode holders. The furnace foundation is 2 feet 6 inches above the main floor of the foundry. The whole furnace is incased in plate steel ⅝ inch thick.

In lining the furnace a mixture of tar and dolomite is tamped in around a sectional mold to a depth of 9 inches to form the bottom. The sides of the hearth slope about 60 degrees up to about 18 inches above the bottom. From here up to the roof, 15 inches, the lining is magnesite brick. The lining is 12 inches thick on the sides and 14 inches thick on the ends. The internal dimensions of the furnace are 4 feet 2 inches by 1 foot 8 inches at the bottom of the hearth, and 5 feet by 2 feet 7 inches at the top, with a depth of 31 inches. The doors are 9 by 10½ inches. The customary roof for this type of furnace is silica brick, set so as to give a roof 12 inches thick. At Braintree considerable difficulty has been experienced with the roof and several kinds of brick have been tried, such as silica, magnesite, and bauxite. Bauxite brick were being used recently.

The electrodes are threaded for continuous feeding and are used in either 4 or 6 foot sections. Each of the two electrode holders consists of two vertical 9-inch channel irons that are set 5 inches apart and act as guides for two ⅜-inch copper plates. These vertical copper plates are attached to a heavy horizontal copper plate that extends over the furnace and has a clamp tightened by a screw for supporting the electrodes. The electrodes may be raised or lowered by hand wheels at the rear or by automatic Thury regulators. In later furnaces the copper, that serves to support the electrode as well as conduct the current, is to be replaced by manganese steel supports and copper bus bars large enough to conduct the current. This will strengthen the holder and reduce the amount of copper, about 1 ton being necessary in the old design. The electrode holders are water cooled at the electrodes, and there are copper water jackets around the electrodes where they enter the roof.

The cost of a 2½-ton Héroult electric steel furnace may be estimated as follows:

The furnace, automatic regulators, platform, transformers, switchboard, and other equipment will cost when completely installed about $12,000 to $15,000, exclusive of crane or buildings; allow an engineering fee of $5,000, and $350 per month and expenses for an expert to establish the operation of the furnace, making a total of about $25,000. The royalty on tool steel produced would be from $1 to $3 per ton according to quantity, and on castings, billets, or ingots 50 cents per ton.

THREE-PHASE HÉROULT FURNACE.

With the use of three-phase electric current in the Héroult furnace of 15 tons capacity a slightly modified design has been made. Until

recently the largest three-phase Héroult furnaces were the two in this country, one at South Chicago, Ill., and the other at Worcester, Mass., which are of identical design, but recently there has been erected a 25-ton three-phase Héroult furnace at Bruckhausen, Germany, and a 22-ton furnace is almost completed at the same plant.

The essential difference between the 15-ton, three-phase furnace at South Chicago, Ill.,[a] and the single phase 2.5-ton Héroult furnace is that in place of two electrodes there are three let down through the roof so that their ends form the vertices of an equilateral triangle, each side of which is 5 feet 2 inches long, one vertex of this triangle pointing toward the back of the furnace. The center of this triangle coincides with the center of the furnace. Each electrode is connected to a phase of a three-phase circuit.

A steel overhead structure supports the electrodes, the weight being directly supported by chains which run back over pulleys on the framework to the drums at the back of the furnace. The electrodes are kept in alignment by vertical guides. The chains are attached to three separate solid copper holders, which are bolted directly to the bus bars. In front these holders are split and joined with a right-and-left screw, which enables the holder to be opened or closed at will. The holders can be made to carry any size of electrode up to 24 inches diameter by use of contact blocks.

The electrode may be regulated by three hand regulators about 4 feet back of the furnace, or by automatic regulators. There is an individual motor for each electrode and a complete automatic device similar to the Thury regulator.

The furnace proper has a shallow hearth, as in the single-phase furnace, but is circular instead of rectangular. The outside dimensions are approximately that of a complete circle 13.5 feet in diameter, with two flattened portions at the front and back. The furnace shell is of plate steel 1 inch thick, riveted together.

The furnace bottom is made of one row of magnesite brick laid on edge across the steel shell, over which is rammed dead-burned Spaeter magnesite to a depth of 12 inches at the center—its thinnest point. The side walls consist of two rows of magnesite bricks laid on end, giving a thickness of 18 inches up to the furnace roof. The roof is made of silica brick and is 12 inches thick. There is an 8-inch rise in the 10-foot span across the furnace.

The furnace has five doors, two on each side and one in front over the pouring spout, of cast iron lined with fire brick, 4½ inches thick. They are operated by steam pressure, with the exception of the one over the pouring spout, which is operated by hand with a counterbalance.

[a] Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 205.

The foundation is of concrete and extends 5 feet above the ground. On the foundation is a stationary rack 8 feet 9 inches long, upon which the furnace proper rests on a floating pinion fastened to the shell by rivets. The arc of this floating pinion has a radius of 10 feet, which gives the furnace a tilting angle of 29°. Attached to the extreme back of the furnace is an 18-inch plunger with a 4-foot stroke working in a cylinder attached to a hydraulic line of 500 pounds pressure per square inch, which gives a lifting power of about 45 tons. The furnace rights itself by its own weight after tilting.

For operation the furnace takes 1,200 to 1,500 kw., supplied by a three-phase current at about 90 volts, and having a frequency of 25 cycles.

THE GIROD FURNACE.

The main difference between the Girod electric steel furnace and the Héroult furnace is that the former has a hearth which conducts the electric current. There are in operation 16 Girod furnaces of 2.5 to 12 tons capacity and 5 are in course of construction. This type of furnace seems to be especially satisfactory in the refining of cold scrap steel because of the slight fluctuations in power demand.

SINGLE-PHASE GIROD FURNACE.

The single-phase furnace at Ugine, France[a] (fig. 28), has a shallow conducting hearth of dolomite with pieces of soft steel embedded in the dolomite near the periphery, and a carbon electrode passes through the roof. In the operation of the furnace current passes through the carbon electrode and through the steel bath, which touches the tops of the steel poles embedded in the hearth. The furnace is mounted on rollers and tilted by an electric motor. This furnace has a capacity of 2.5 to 3 tons and is operated with 300 kw. The voltage is 60 to 65 volts, and the frequency of the current is 25 cycles.

The hearth of the furnace is 3 feet square at the bottom and 6 feet square at the top. The roof is 31 inches above the hearth. The water-cooled steel electrodes (fig. 29), embedded in the hearth when new, project beyond the bottom of the hearth a short distance. There are 6 steel poles set on the circumference of a 31-inch circle. The floor of the hearth is made of dolomite and tar rammed in. The walls are magnesite brick; the roof is silica brick. The roof is insulated from the walls by a thin layer of asbestos. The whole hearth is encased in a $\frac{5}{8}$-inch steel shell and is set on a concrete foundation that extends 6 feet above the ground level. The furnace has one

[a] Borchers, W., Electric smelting with the Girod furnace: Trans. Am. Inst. Min. Eng., vol. 41, 1910, p. 120.

charging door at the rear and a pouring spout in front, and resembles the open-hearth furnace even more than the Héroult furnace does.

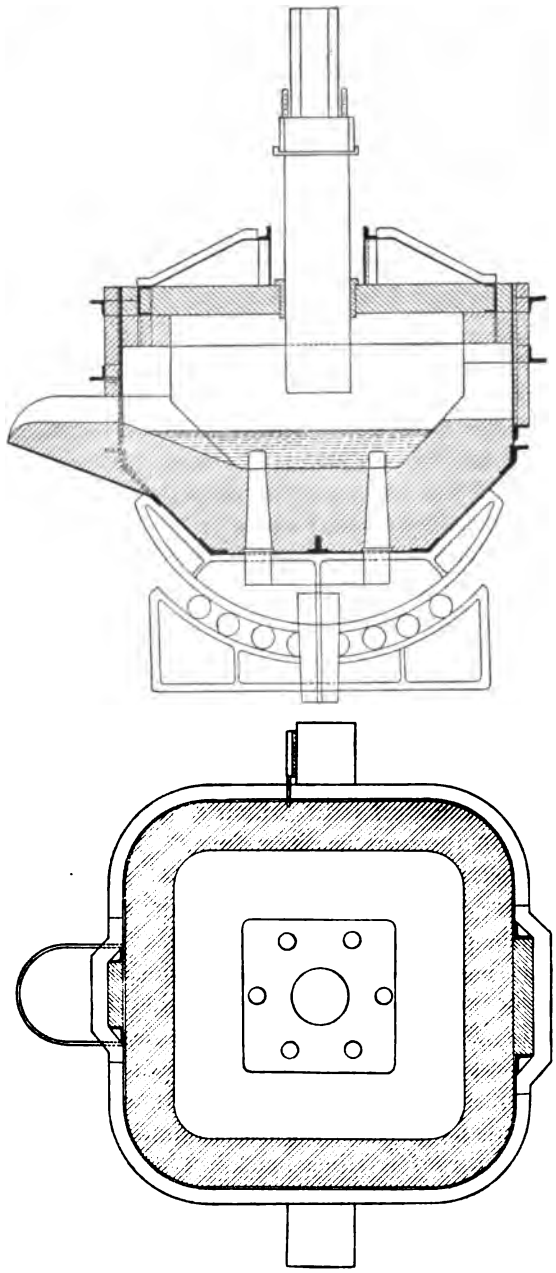

FIGURE 28.—Plan and elevation of 2.5 to 3 ton, single-phase Girod steel furnace, Ugine, France.

The 2.5-ton furnaces at Ugine are operated with carbon electrodes 14 inches in diameter and 5 feet long, some of which are threaded for continuous feeding. The means of supporting the electrode differs from the Héroult scheme in that the supporting steel structure is built over the furnace from the sides (fig. 22), making it possible to have a door at the rear. The crosspiece holds a water-cooled holder attached to the electrode. There is also a water jacket around the electrode where it passes through the roof. The crosspiece is raised or lowered by a screw at each end, the side structural work serving as a guide. The electrode may be adjusted by hand or by automatic control.

With a conducting-hearth furnace there is a considerable tendency to the presence of induced currents in the steel shell which reduce the power factor. Hence the arrangement of the electrical conductors is of great importance in the Girod furnace. Three meth-

ods have been used: (1) The shortest path from the motor-generator set to the carbon and steel electrode is used, so that all of the cables are on the side of the furnace that faces the motor generator. (2) The cable to the electrode is in two parallel sections, and the steel electrode is connected by the shortest path to the motor-generator set, so that each steel pole has a direct cable connection with the generator. (3) The third method is now being adopted for the latest furnaces of this type. The current is conducted to the carbon electrode in a manner similar to method 2, but whereas in methods 1 and 2 the bottom steel electrodes are insulated from the furnace body, in this method the steel electrodes are electrically connected to the furnace body; also the conductors are bus bars instead of cables attached to the steel shell of the furnace. These bus bars are arranged symmetrically around the furnace. This arrangement is shown in figure 22.[a] The advantages of this arrangement are: A better agitation of the bath due to the arc circling around the periphery of the carbon electrode, greater durability of roof and lining, a saving of 10 per cent of energy consumption, the use of copper bus bars instead of cables, lack of current interruptions due to rupture of the arc, and reduction of electrode consumption.

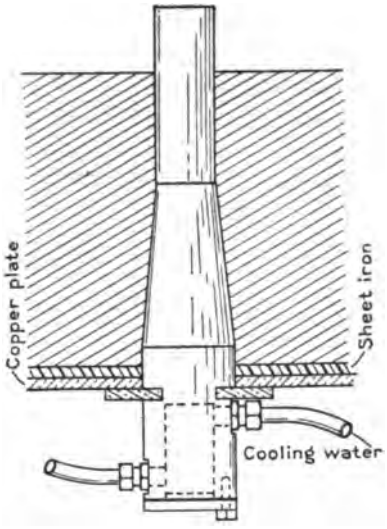

FIGURE 29.—Section of water-cooled steel electrode used in Girod furnace.

The cost of the metallic parts of a 2.5-ton Girod furnace, including electrode regulators, measuring instruments, tilting device, and conductors from the furnace to a dynamo or transformer near the furnace room, not including transforming or generating machinery and license fee, is estimated at about $3,000. The cost of a plant consisting of one 2.5-ton furnace for regular running and one furnace for reserve, with all appliances and smelter building, but without dynamo or transformer, is estimated to be approximately $40,000 to $50,000. The license fee is not included in this estimate.

THREE-PHASE GIROD FURNACE.

A section of the three-phase 10 to 25 ton Girod furnace is shown in figure 30. There are four upper carbon electrodes 14 inches in

[a] Mueller, A., The manufacture of steel in the Girod electric furnace: Metall. Chem. Eng., vol. 9, 1911, p. 581.

diameter, which with their connections constitute the essential difference between the single-phase and the three-phase furnace. On a three-phase circuit the Girod furnace is connected by the star con-

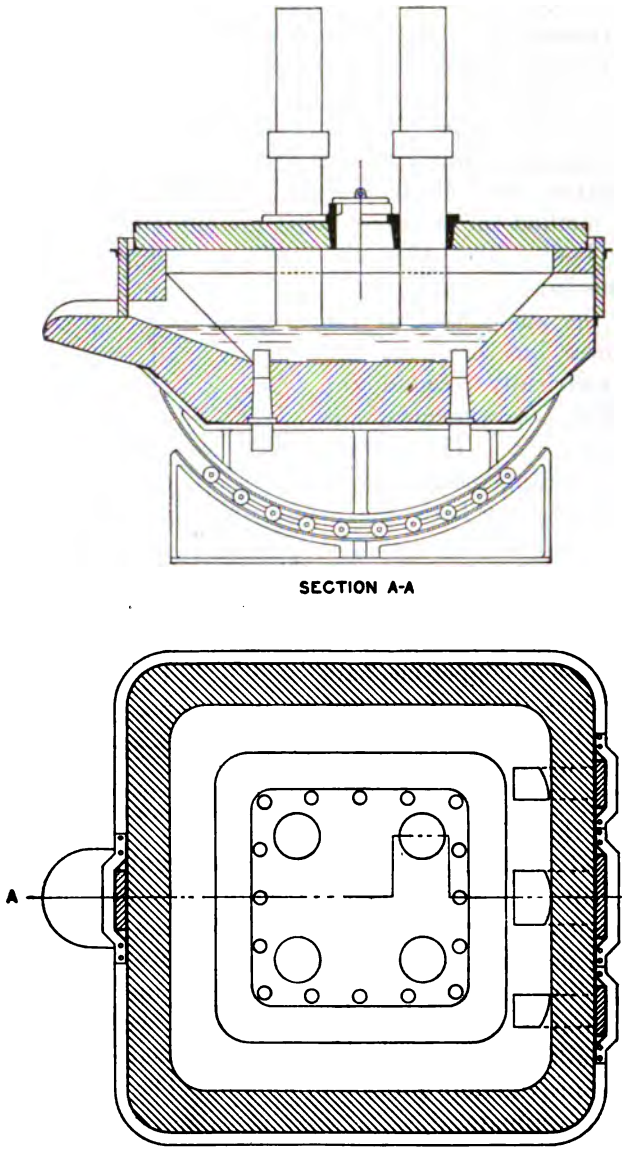

FIGURE 30.—Plan and elevation of 10 to 12.5 ton, three-phase Girod steel furnace, Ugine, France.

nection. Two of the carbon electrodes are each connected to a phase. The other two are connected in parallel with the third phase, while the hearth is connected so as to form the neutral point of the system.

There are 16 bottom electrodes of steel. The 12-ton furnace uses from 1,000 to 1,200 kw., supplied by a 25-cycle current at 70 to 75 volts.

The interior of the hearth of the 12-ton furnace is 4 feet square at the bottom, widening out to 10 feet at the top. The silica brick roof is set 3 feet 9 inches above the hearth of dolomite. The bottom is tamped in to a depth of 20 inches.

The furnace, with the exception that there are three charging doors at the rear instead of one, is similar to the single-phase 2.5-ton furnace. The four electrode holders are set two on a side, arranged as in the smaller furnaces.

The metallic parts of a 10 to 12.5 ton three-phase Girod furnace, including regulators for the electrodes, measuring instruments, tilting mechanism, and conductors from the furnace to a dynamo or transformer near the furnace room, but not transforming or generating machinery and license fee, cost about $6,000. A plant with one 12-ton furnace in reserve, including building, but without dynamo or transformer, is estimated to cost, license fee excluded, about $60,000 to $100,000.

THE STASSANO FURNACE.

The Stassano[a] electric steel furnace differs from other electric steel furnaces in that the electrical current does not pass through the metal or slag. All heating is by radiation from three horizontal arcs. The power consumption of the Stassano furnace is somewhat higher than that of some others, so that its use is limited to furnaces having a capacity of not more than 2 tons for making high-grade small steel castings. At present there are 16 Stassano furnaces in operation in sizes up to 2 tons capacity and one is in course of construction.

The Stassano furnace (figs. 19 and 20) has a circular hearth with a cylindrical melting chamber above. In the 1-ton furnace (figs. 19 and 20) the hearth is about 3 feet in diameter, with the roof 3 feet 6 inches above it. The furnace is incased in a steel shell with one door opposite the pouring spout. There are also openings for the three electrodes which project toward the center of the melting chamber. The movement of each electrode is controlled by a hydraulic piston. The source of power is connected by a rod with the end of each electrode, each of which is connected to a phase of a three-phase system. Each electrode holder and electrode are surrounded by a water jacket. In the 2-ton furnace carbon electrodes 3 inches in diameter and 4 or 5 feet long are used. The lining of the hearth is dolomite and tar, but the roof and sides are magnesite

[a] Stassano, E., The application of the electric furnace to siderurgy: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 63.

brick. The external dimensions of the 1-ton furnace are about 10 feet by 8 feet; the 2-ton furnace is 10 feet high and 10 feet in diameter.

The early type of Stassano furnace had a chamber beneath it (fig. 19) which contained a rotating mechanism for agitating the steel bath. This feature has not proved of great value, and is now abandoned in the more recent furnaces, which, like the Girod furnace, are set on rollers. Some furnaces recently erected at Newcastle on Tyne, England, by the Electroflex Steel Co. have a combination of the rotating and tilting motion.

A 1-ton Stassano furnace requires 200 kw. supplied by a 3-phase at 110 volts, having a frequency of 25 cycles. A 2-ton furnace uses about 400 kw. at 120 volts.

THE KELLER FURNACE.

The Keller[a] furnace (fig. 23) is very similar to the Girod furnace, as it has a conducting hearth of iron rods embedded in a refractory material. This type of furnace was the first to be used as a superrefining agent for steel that had been made by the old-established methods. There are in Europe three Keller furnaces, the capacities of which are 1 to 8 tons.

The Keller furnace consists of a conducting hearth surrounded by a steel water jacket, with a silica brick roof. The conducting hearth consists of iron bars 1 to 1½ inches diameter, set vertically 1 inch apart in an iron plate. These bars are surrounded by a mixture of magnesite and tar rammed in while hot. The bars are good conductors of electricity when the furnace is cold, and the magnesite also becomes a conductor when hot. The small furnaces have one carbon electrode and the larger ones have four electrodes. In both types the connections are similar to those of the Girod furnace. A feature of the Keller furnace plant, devised before the day of continuous feeding of electrodes, is the revolving arms, with extra electrodes for quick charging (fig. 23). An 8-ton Keller furnace is operated with 750 kw.

THE GRÖNWALL FURNACE.

In general external appearance the Grönwall[b] or Electro-Metals furnace (figs. 24 and 25) resembles the single-phase Héroult furnace. However, it uses two-phase current and has a conducting hearth. A noteworthy feature of this design is the steadiness of the load on the power line when cold scrap is worked, as the arcs are not connected in series. There are four Grönwall furnaces in operation.

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[a] Keller, C. A., A contribution to the study of electric furnaces as applied to the manufacture of iron and steel: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 87.

[b] Robertson, T. D., The Grönwall steel refining furnace: Metall. Chem. Eng., vol. 9, 1911, p. 573.

The current may be directly supplied as two-phase or transformed from three-phase to two-phase by the Scott connection of transformers. In the Electro-Metals furnace at Sheffield there are two 12-inch carbon electrodes projecting into the furnace through the roof, each of which is connected to a phase, the conducting hearth forming the neutral of the system. This conducting hearth consists of carbon paste rammed on the steel bottom of the furnace to a depth of 4 inches, over which is placed a mixture of dolomite and tar to a depth of 10 inches. In figures 24 and 25 the neutral point is a carbon block, but, as stated above, this has been recently changed at the Sheffield furnace. By this connection half of the power goes through one electrode and half through the other, each being independent of the other.

The furnace proper is rectangular in shape, like the Héroult furnace, the exterior being 6 feet 6 inches wide by 8 feet 10 inches long by 4 feet 11 inches high. There are two doors at the sides and a pouring spout, with the electrode holders, at the rear. The holders are of manganese steel, held tightly around the electrodes by wedges. At first, as shown in figure 25, current was led to the electrodes through these holders, but at present the electrodes are electrically connected with cables and two copper plates, $\frac{1}{4}$ inch thick and 6 inches wide, shaped so as to pass around the electrode. The furnace is tilted by hand by means of a screw device. Electrodes are adjusted by hand, although there is no technical reason why an automatic regulator should not be used. The lining of the sides is magnesite brick and the roof is silica brick. The use of water jackets around the electrodes has been discontinued. As now operated there is no water cooling of any part of the furnace. The furnace is set upon a concrete foundation, being tilted on the usual curved bars. The power required is 500 kw., the frequency of the current being 25 cycles; the voltage of the two carbon electrodes is 105, while between each electrode and the neutral point, the hearth, the voltage is about 75.

THE NATHUSIUS FURNACE.

The Nathusius[a] furnace (fig. 26) is three-phase and combines heating from above by arcs with resistance heating from below in the steel bath for the purpose of decreasing local heating by the arcs. The furnace is circular in form and is incased in a steel shell. There are three water-cooled carbon electrodes which project through the roof into the furnace above the surface of the charge, and three or a multiple of three water-cooled bottom electrodes of mild steel set in the hearth. Both upper and lower electrodes are arranged in a

[a] Nathusius, H., The refining of steel in the Nathusius electric furnace: Jour. Iron and Steel Inst., vol. 85, 1912, No. 1, p. 51.

triangle (p. 76). The electric connections are shown in figure 26. The electrodes are suspended by cables from overhead runways and are adjusted either automatically or by hand. On tilting, these electrodes are raised out of the furnace. The tilting mechanism is operated by an electric motor. Furnaces under 6-ton capacity are on trunnions, but with the larger sizes rollers are used. The 5-ton furnace at Friedenshütte, Germany, is normally operated with about 600 kw. Sixty-cycle current is used. The voltage between the upper electrodes is 110 volts, between the lower electrodes 10 volts, and between the upper and lower electrodes 61 volts. In addition to this 5-ton furnace the same company operates a 2 to 3 ton furnace for melting ferro-alloys.

OTHER ELECTRIC STEEL FURNACES OF THE ARC TYPE.

Several other types of arc furnaces which have no especially novel feature are in operation at the plants where they were originally designed. The Chaplet furnace, used for the manufacture of ferro-alloys, the direct production of steel from ores and scrap, is similar in principle to the Héroult furnace, but has two separate chambers with an electrode in each. Four Chaplet furnaces are in operation in France. The Anderson furnace is also similar to the Héroult furnace, but has an electromagnet beneath it for the purpose of controlling the position of the arc. Five Anderson furnaces have been built in England. The Stobie two-phase furnace is very similar to the Grönwall furnace. There has also been designed a Stobie three-phase furnace. Four Stobie furnaces are being built at Newcastle, England. One three-phase Soderburg furnace, similar to the Girod furnace, is in operation. In the design of the Harden Paragon[a] furnace the bath is heated from above by arcs, and also from the sides and bottom by side plates in the lining.

INDUCTION FURNACES.

THE KJELLIN FURNACE.

The Kjellin furnace, the pioneer of induction steel-furnaces, is especially adapted to the melting of fine materials to obtain a high-grade steel. At present 9 of these furnaces are in operation, but the writer does not know of any others being erected.

The Kjellin[b] furnace (fig. 21) is in reality a transformer in which the bath of molten metal forms the secondary circuit. The magnetic

[a] Hårdén, J., The Paragon electric furnace and recent developments in metallurgy: Met. and Chem. Eng., vol. 9, 1911, p. 595.

[b] Kjellin, F. A., The Kjellin and Röchling-Rodenhauser electric furnaces: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 173.

circuit C is built up of laminated sheet iron like the core of a transformer. The primary circuit is a coil consisting of a number of turns of insulated copper wire or tubing surrounding the magnetic circuit. The ring-shaped crucible B, made of suitable refractory materials, also surrounds the magnetic circuit, and when filled with molten metal forms the secondary circuit of the transformer. The annular crucible is supplied with covers.

If the coil be connected with the poles of an alternating current generator, the current passing through the coil excites a variable magnetic flux in the iron core, and the variation in the magnetic flux induces a current in the closed circuit formed by the molten metal in the crucible B. The ratio between the primary and secondary current is fixed by the number of turns of the primary, and the magnitude of the current in the steel is then almost the same as the primary current multiplied by the turns of the primary coil. Thus in a small furnace of this type a current of 500 volts and 280 amperes supplied to the coil induces a current of 7 volts and 24,000 amperes in the metallic bath. Before starting the furnace an iron ring must be placed in the crucible and melted down to form a bath, or the crucible must be filled with molten metal taken from another source. On continuous work it is customary to leave enough metal in the crucible to establish the bath. Kjellin furnaces have been built in sizes up to 8.5 tons capacity, requiring 750 kw.

THE RÖCHLING-RODENHAUSER FURNACE.

The Röchling-Rodenhauser furnace is an improvement of the Kjellin induction furnace designed especially for the refining of molten basic Bessemer steel and for use with currents of frequency ordinarily used in steel plants. The Kjellin furnace of 8 tons capacity required a current of not more than 5 cycles frequency or the power factor would drop below 0.6. The Röchling-Rodenhauser furnace is so designed as to have a power factor of 0.6 when using a frequency as high as 50 cycles. The furnace is the most widely used of the induction furnaces and is especially adapted to refining molten metal. Eighteen of these furnaces are already in operation and four are in course of construction.

The Röchling-Rodenhauser[a] furnace (fig. 27) has a hearth of very different shape from the Kjellin furnace, as it has a distinct open hearth which no other induction furnace possesses. Both single and three-phase current furnaces are constructed, the former having two grooves in which the metal is melted and the latter three. In either

[a] Rodenhauser, W., The electric furnace and electric process of steel making: Jour. Iron and Steel Inst., vol. 79, No. 1, 1909, p. 261.

furnace these grooves or heating channels, each of which corresponds to the annular hearth of the Kjellin furnace, open into a distinct open hearth, where all metallurgical operations, such as the addition of fluxes and alloys, take place. The grooves, which have a comparatively small cross section, form the secondary circuits in which the currents that heat the metal are induced.

Two side doors are provided in a single-phase furnace and three in a three-phase furnace. The furnace is set on rollers and is tilted by means of a hydraulic motor. One door has a spout for pouring. As the doors are slightly above the level of the bath, removing the slag is no more difficult than in the arc furnace, which is not the case with other induction furnaces.

In figure 27, HH represents the two legs of the transformer that are surrounded by primary coils A connected with the alternating-current circuit. Secondary currents are induced in the two closed circuits formed by the bath. These two circuits are connected by the hearth in the center so as to resemble the figure 8. The primary coils are so arranged that the induced currents have the same direction in the common part of the two circuits. The difference between this furnace and the ordinary induction furnace consists in the use of extra secondary coils BB surrounding the primary coils AA. The secondary coils are connected to metallic plates EE, covered by an electrically conducting mixture of lining material, G, which forms part of the lining of the furnace. The current from the secondaries passes through the plates, E, through the lining, G, and then through the main hearth, D.

This current used in combination with the current from the channels CC gives a better power factor than when the furnace is operated with the current from the channels only. Another advantage is that the magnetic leakage field surrounding the primary coils, which formerly had the effect of checking the primary current and decreasing the power factor, is now utilized for inducing currents in the extra secondaries.

The result is that the main hearth can be made of much larger cross section than before and that, nevertheless, even in big furnaces a good power factor can be maintained without the use of such a low frequency of the current as was necessary with the original induction furnace.

The single-phase furnace gives better satisfaction than the three-phase furnace because it is less complicated. Also, the strong circulation of the bath in the central hearth of the three-phase furnace causes great wear on the lining.

Röchling-Rodenhauser furnaces have been built in various sizes. The 8 to 10 ton single-phase furnace at Volklingen, Germany, takes 600 kw., supplied by a current of 4,000 to 5,000 volts with a fre-

quency of 25 cycles. The 2 to 3 ton 3-phase furnace at the same place takes from 200 to 250 kw., supplied by a 400-volt current with a frequency of 50 cycles. The power load of these furnaces is very steady. The 2 to 3 ton furnace recently installed at Landsdowne, Pa., requires 300 kw., furnished by a 25-cycle current at 480 volts.

OTHER INDUCTION STEEL FURNACES.

The Frick furnace is very similar to the Kjellin furnace, differing from it only in slight details of design. The primary windings are placed above and below the annular ring instead of within it. One 12-ton Frick furnace is in operation in Germany.

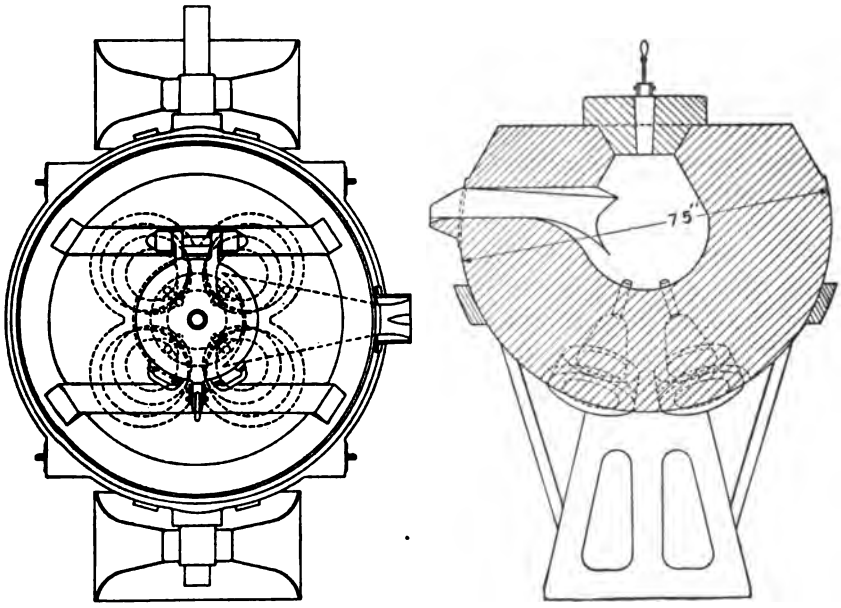

FIGURE 31.—Plan and elevation of Hering resistance furnace.

The Hiorth furnace is a further development of the Kjellin furnace, differing only in that the bath lies in two annular grooves instead of one, thus surrounding both sides of the primary coil. One tilting Hiorth furnace has been erected in Norway.

AN ELECTRICAL RESISTANCE FURNACE ADAPTABLE TO STEEL MANUFACTURE.

A resistance furnace based upon the "pinch effect" has been designed and operated on an experimental scale by Hering.[a] Although this furnace (fig. 31) is not in commercial operation as

[a] Hering, C., A new type of electric furnace: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 255.

yet for the production of steel, several are in the course of construction for this purpose, and small ones have been shown in operation. The following is a brief description of the principle on which this resistance furnace operates.

The "pinch phenomenon" is the local contraction of cross section of a liquid resistor through which electric current is passing and in which heat is being generated. In open channels this contraction or pinching often results in complete rupture, thereby limiting the temperature. This contraction is caused by an electromagnetic force that acts from the circumference to the center of the conductor and in a direction perpendicular to the axis.

If a conductor consists of a column of liquid metal in a vertical hole in a nonconducting material closed at the bottom by the electrode and opening at the top into the bath of metal in the hearth, then this force acts horizontally, and perpendicularly to the axis along the whole length, which results in an axial force that causes the liquid metal to flow out of the center of the open end. In the Hering furnace such a column is made the resistor in which the desired heat is generated by passing the current through the column by means of an electrode at the bottom. Two such resistors are placed in the bottom of a furnace of any desired shape for single-phase current and three for three-phase current. As this "pinch effect" can not now rupture the circuit, it expels the heated liquid rapidly from the holes and forces it against the blanket of slag, thereby continually renewing the surface exposed to the slag action, while the cooler liquid in the bottom of the hearth is sucked down into the hole near the circumference to be in turn heated and immediately expelled. The ejecting force increases as the square of the current and diminishes with an increase in the cross section of the conductor.

There is an active and systematic circulation of the liquid bath, and the temperature is very evenly distributed throughout the bath. As far as is known now, there is no temperature limit except that which causes failure of the refractory lining. The electrodes may be made of metal, and no adjustment of them is necessary; they are not consumed. The hottest liquid is in the center of the resistors. The rapid flow is not along the walls of the hole but in the center. The power factor can be made very high, and ordinary frequencies may be used.

The furnace can be operated with direct or alternating current of one, two, or three phase. The transformers in the latter case are attached directly to the bottom of the furnace, as the amperage is very high.

ELECTRIC STEEL MANUFACTURING PRACTICE.

There are two courses of procedure for steel manufacture in which the electric furnace has been used: Cold scrap iron and steel of either inferior or high-grade quality is melted and refined in an electric furnace with the production of steel of the highest grade equal to the best crucible steel; and molten steel, the product of either the acid or basic converters, or of the acid or basic open-hearth furnaces, is superrefined, or made into alloy steel, in an electric furnace.

MANUFACTURE OF STEEL FROM SCRAP IRON AND STEEL IN THE ELECTRIC FURNACE.

SOCIÉTÉ ÉLECTRO-MÉTALLURGIQUE FRANÇAISE, LA PRAZ, SAVOIE, FRANCE.

The plant of the Société Électro-Métallurgique Française, at La Praz, Savoy, France, has passed through many phases of the use of the electric furnace in metallurgical operations. The first product produced here by Héroult was aluminum, which was followed by calcium carbide, ferro-alloys, electrodes, and steel. To-day the plant uses 7,500 kilowatts for the production of aluminum, steel, and electrodes.

DESCRIPTION OF PLANT.

The principal products of the 2.5 to 3 ton Héroult furnace (fig. 18) in operation at La Praz are high carbon and alloy tool steels. The furnace is the original Héroult furnace examined by the Canadian commission. It differs from the furnace at Braintree in that the lining is thinner, giving a more shallow hearth. The hearth is ground dolomite and tar, the sides are magnesite brick, and the roof is silica brick. The two carbon electrodes are 14 inches square. The holders are of somewhat more simple design than those in the Braintree furnace (p. 94). They consist of two clamps of steel held together by a pin, being tightened by a wedge. Current is brought in by cables attached to pins in the ends of the electrodes and to the electrode holders. Electrodes are not threaded for continuous feeding. The holders and ports in the roof are water cooled. The electrodes may be regulated either by hand or automatically. Power is supplied by a single-phase alternator directly connected to the furnace about 30 feet away. The power required is 300 kw., and is supplied by a 33-cycle single-phase current, at 100 to 110 volts, giving 50 to 55 volts on each electrode.

PRACTICE AT PLANT.

The furnaces are charged with low-carbon steel or wrought-iron turnings and a part of the first refining flux. As the melting proceeds, all of the first flux, consisting of iron ore and lime, is added to remove the phosphorus. When this is accomplished the slag is

completely removed from the furnace and a flux of lime added to remove the sulphur. The slag is completely deoxidized for the removal of sulphur by the addition of coke dust. The metal is finally carburized to the desired point by addition of carburite, and then poured into ingots.

The results of operations covering one week are given below. The number of heats is probably somewhat greater than the usual number, as at the time of the writer's visit the average was three in 24 hours. A noteworthy reduction of power consumption has been made over that of 1904, as the figure of December 24, 1911, was 528 kilowatt-hours per ton as compared to 840 kilowatt-hours per ton, of 1 per cent carbon steel, refined with two slags, for 1904, a reduction of 312 kilowatt-hours, or 37.1 per cent decrease. The electrode consumption of 18 kg. (39.6 pounds) in 1904 would probably approximately hold for to-day, as the methods have not been changed materially in that respect. The roof had stood 107 heats up to the week mentioned.

Results of operation of furnace for week ended Dec. 24, 1911.

Hours worked	126
Number of heats	26
Average hours per heat, including all repairs and charging	4 hours, 51 minutes.
Shortest heat	3 hours, 10 minutes.
Total weight teemed	66 tons.
Lowest power consumption, per ton, at furnace	459.2 kilowatt-hours.
Average power for week at furnace, per ton	528 kilowatt-hours.
Scrap, per cent	3
Clear ingots, per cent	93
Loss by oxidation, per cent	4

FINAL FORM OF PRODUCT.

The ingots cast at the furnace are reheated and reduced in rolls and hammered to 1 inch to ½ inch rods of circular or square cross section. The product is sold in this form. Samples of steels of the following composition were taken by the writer:

Tool steels produced in Héroult furnace, La Praz, France, 1912.

	No. of sample.			
	1	2	3	4
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Carbon	0.904	1.034	1.196	0.990
Silicon	.323	.201	.170	.700
Manganese	.236	.273	.210	.211
Sulphur	.008	.008	.009	.024
Phosphorus	.004	.006	.012	.007
Tungsten				19.407
Chromium				6.717

PLANT OF MESSRS. VICKERS, LTD., SHEFFIELD, ENGLAND.

A 2.5 to 3 ton single-phase Héroult furnace, similar to the furnace at Braintree, is being used in the manufacture of steel of crucible grade from cold scrap iron and steel at the River Don plant of Messrs. Vickers, Ltd., Sheffield, England. A new 8-ton three-phase Héroult furnace is being constructed, which will be directly connected to the main power circuit of the works and will take about 1,200 kw. The lining of rammed dolomite and tar, both on sides and bottom, will be thinner than usual, so that the nominal capacity will be 10 instead of 8 tons. The electrodes are to be supported by holders projecting over the roof from the sides. Charging doors will be placed in the rear instead of at the side, with the pouring spout at the front. The furnace will be set close to the edge of the working floor instead of 10 feet back, as is the case with the present 2.5-ton furnace. The erection of an electrode plant is also being considered.

DESCRIPTION OF PLANT.

The electric furnace, generator, and laboratory are placed in a steel frame building about 60 feet square, covered with corrugated iron. In one corner, at the rear of the furnace on the ground level, is the generator room. Above, on the furnace level, are the office and laboratory. The furnace is set on a concrete foundation extending about 6 feet above the ground with a steel platform about 20 feet square around it. The furnace is at the rear of this platform, leaving an open space of about 10 feet in front, in which there is a door that can be lifted for setting the ladle beneath the spout in pouring. The instrument board and Thury regulators are back of the furnace. In front of the platform is a pit 5 feet long, 20 feet wide, and 4 feet deep, where ingot molds are set for teeming. The remaining floor space is devoted to repairing and heating ladles, storing ingots, and piling scrap. There is an electric crane in front of the furnace platform.

The furnace differs in some details from the Braintree furnace. The bottom consists of a 4½-inch layer of magnesite brick laid on edge over the steel shell, above which tar and dolomite are rammed in to a depth of 7 inches, making a hearth 11.5 inches thick. This slopes up to a line on the sides above the slag line. When the furnace was first operated the walls were built of magnesite bricks laid to give a thickness of 9 inches. Because of the bricks spalling they have now been laid so as to give a wall 4½ inches thick. This, incidentally, has increased the capacity, so that the average charge is now 3 tons instead of 2.5 tons. The roof is silica brick. Carbon electrodes 16 inches in diameter and 4 feet long, and threaded for continuous feeding, are used. The electrodes are supported by a more simple holder than that of the Braintree furnace, consisting of bronze clamps surround-

ing the electrode, tightened in front by a horizontal bolt. The holder is similar to the older type, except for this feature, and that it is made entirely of manganese steel instead of copper and is not used for conducting current. Current is conducted by means of copper bus bars 4 inches wide and ⅜ inch thick from cables at the rear of and beneath the furnace to the bronze holders around the electrodes. The furnace is tilted by means of a screw device that is operated by an electric motor. The electrode holders are water cooled, and there are copper water jackets around the electrode openings in the roof.

Power is delivered to the furnace about 20 feet from the generator by a single-phase alternating-current generator of 600 kw. capacity, operating at 100 to 110 volts and a frequency of 25 cycles. The alternator is directly connected to a 3-phase motor. The power is received from the main power plant of the works at 2,200 volts. The current consumption charged to the furnace is that shown on the primary side of the main 3-phase line, so that it includes all transforming and other losses. In the generator room there are also a power-factor meter and other meters on the secondary side. Current is conducted to the furnace bus bars by insulated copper cables. At the rear of the furnace are the Thury regulators, a wattmeter, a voltmeter on both electrodes and on each single electrode.

PRACTICE AT PLANT.

The electric-furnace operation consists in making high-grade steel for various purposes, such as tools and armament, out of scrap iron and steel turnings, some of which are impure ordinary carbon steel, while others contain valuable proportions of tungsten, chromium, and other expensive alloys. The furnace is operated at 100 volts, 50 volts per electrode, with a power consumption of 400 to 550 kw. For 15 to 45 minutes after charging, the electrodes are regulated by hand, but as soon as a pool of molten metal has formed in the bottom the Thury regulators are thrown in. The constant breaking of the circuit is very noticeable before the charge is completely melted. Sometimes one and sometimes two refining fluxes are used, depending upon the impurities present. The first flux is generally 40 kg. (88 pounds) of lime, 30 kg. (66 pounds) of hematite, and 20 kg. (44 pounds) of fluorspar. The second flux consists of 40 kg. (88 pounds) of lime and 20 kg. (44 pounds) of fluorspar, and is added only after complete dephosphorization by the first slag. To deoxidize the second slag for removing sulphur, coke dust is added. Sometimes pig iron, ferrosilicon, ferromanganese, and sand are used at the end of a run. Aluminum is thrown in the ladle bottom before pouring. A 3-ton ladle is used for teeming and is preheated before using. Part of the metal from one heat is cast by bottom casting

into five 500-kilogram (1,100-pound) ingots, the balance of the charge being cast by top casting into 100-kilogram (220-pound) ingots. Of the metal charged, about 85 per cent is recovered in good ingots, 5 per cent in poor ingots, 3 per cent in scrap, and 7 per cent is oxidized.

Working with high-speed tool-steel scrap, the method of running a charge was as follows:

Results from a single charge.

Charge, steel scrap, tool-steel scrap, 2,800 kg. (6,160 pounds).
Slag added, 40 kg. (88 pounds) lime and 20 kg. (44 pounds) fluorspar.
No slag skimmed.
Additions at end, 6 kg. (13.2 pounds) of a deoxidizing mixture, 2 kg. (4.4 pounds) ferrosilicon, 0.5 kg. (1.1 pounds) aluminum. No coke dust added.
A gray slag.

Time, hours	3½
Total power on primary meter, kw	1,580
Kilowatt-hours per metric ton of sound ingots	542
Average power, kw	513

Composition of product:

	Per cent.
Carbon	0.60
Tungsten	4.57
Chromium	5.02
Manganese	.20
Silicon	.07
Phosphorus	.012
Sulphur	.052

It will be noted that the power consumption is higher in this case for an operation with one slag than at La Praz with two slags, but this may be explained by the fact that power charged to the furnace at Sheffield is read on the primary circuit and includes all losses; also results are figured to tons of sound ingots. The power factor averages about 0.85, but varied over a period of a year from 0.79 to 0.95.

The average length of a heat is 4 hours, but at times as many as 6 heats per 24 hours have been made, including charging and pouring. The electrode cost is about 75 cents per metric ton which, with electrodes at 4 cents a pound, gives a consumption of 18.7 pounds per ton. It takes about 20 minutes to add a new 4-foot section to an electrode already in the furnace. The electrodes wear uniformly around the holder and at joints, but if a joint is not properly made the electrode tends to wear to a point, and then must be broken off at the end. If a piece falls into the bath it is necessary to shut off the power while the piece is being removed. Between heats the power is off for 10 to 15 minutes. The furnace lining is then fettled with dolomite. For a period of six months' continuous operation the roofs lasted on the average 78 heats, but some have lasted as long as 120 heats.

Advocates of the conducting-hearth type of furnace have stated that it would be very difficult to melt out the charge of a Héroult furnace if frozen, but this does not appear to be the case. At this plant a 2.5-ton charge was frozen, owing to a breakdown of the generator, and stood cold for four days. The charge was melted in 12 hours with a power expenditure of 1,950 kilowatt-hours per ton. During the melting 0.2 ton more of metal was added, making 2.7 tons in the furnace. Of this 2.525 tons were tapped. For about 1½ hours the power on the furnace was only 250 kw. in order to heat the roof gradually.

For the complete operation of this plant seven men and two boys are necessary, as follows: A superintendent, assistant superintendent, melter, assistant melter, ladle man, floor man, one general laborer, and two boys in the laboratory.

LAKES & ELLIOT FOUNDRY, BRAINTREE, ENGLAND.

At the foundry of Lakes & Elliot, Braintree, England, a 2.5-ton, single-phase Héroult furnace is used in making steel of low carbon content for small castings for the automobile trade.

DESCRIPTION OF PLANT.

The furnace, which has already been described (p. 74), is placed in the main casting building of the foundry, close to one wall. At the rear in an adjoining building is a separate power plant for the furnace, consisting of a gas producer, a Westinghouse gas engine, and a single-phase 300-kilowatt generator which delivers a 25-cycle current to the furnace about 20 feet away at 110 volts. A 5-ton overhead electric crane is used for pouring and other purposes.

PRACTICE AT PLANT.

The scrap iron and steel used consists of a mixture of horseshoes, castings, foundry scrap, boiler punchings, and miscellaneous heavy steel scrap of small dimensions. Some pig iron is also used, which is placed between the electrodes and is claimed to assist in the steady operation of the furnace. The steel charged contains from 0.05 to 0.14 per cent carbon. A part of the iron ore and lime that form the dephosphorizing slag is laid upon the floor of the furnace before charging the scrap. This seems to protect the hearth from corrosion. The remainder of this flux is added during the melting period. After skimming the slag a desulphurizing flux of lime and fluorspar is used. The carbon is reduced to 0.06 per cent and recarburization is accomplished by means of powdered carbon. Some ferromanganese, ferrosilicon, ferrotitanium, and aluminum are added at the end of the run or upon pouring. The charge is poured into a 2.5-ton ladle,

from which it is teemed into small shanks for casting into the molds, thus eliminating slag. The electrodes are regulated by hand for 1½ hours after starting the furnace until the bath is partly melted.

Owing to the thick lining used, charges of not over 2 tons can be made, and the average melt is not over 1.5 tons. The melt takes 4.5 to 11 hours, with an average of about 6 hours, depending upon the nature of the charge. Definite figures as to power consumption could not be obtained, but with 300 kw. on the furnace, it is probably from 800 to 1,000 kw. per ton. One charge is run on the day shift and one on the night shift, the furnace standing idle in the meantime. By this arrangement the furnace is empty 2 to 4 hours between melts.

The electrode consumption could not be obtained definitely, but two 5.5-foot sections of 14-inch electrodes will last for 14 melts. These sections weigh about 600 pounds, which with an average charge of 1.5 tons would give a consumption of 25.5 kg. (56 pounds) per ton. The cost of electrodes is about 3.95 cents per kg. (1.8 cents per pound) f. o. b. plant, or 4.95 cents per kg. (2.25 cents per pound) laid down at Braintree.

Considerable difficulty has been experienced at Braintree in getting a durable roof. Several kinds of brick, such as silica, bauxite, magnesite, and a mixture of magnesite and bauxite, have been tried. At present bauxite brick is being used. The roof lasts 14 to 28 heats, depending upon the nature of the charge. The walls are also attacked by the arc breaking against the sides after the charge has been melted. This, however, does not cause so much trouble when a slag is on the bath. Whenever the roof gives out the furnace is entirely relined.

In addition to these difficulties, the Thury regulators did not appear to be giving satisfaction, because they were continually getting out of adjustment. This was the only plant visited where this trouble was mentioned. It was stated that in building a new furnace automatic regulators would not be used, as there was no saving in labor and an increase in first cost by their use.

The effect of producing low-carbon steel upon the working of the furnace was plainly visible at this plant. At the Sheffield plant, which was operating almost entirely on medium or high carbon steel, the power consumption, electrode consumption, and wear on the lining were much lower than at this plant. This was due chiefly to the low-carbon product requiring a longer period in the furnace, and also because in making steel castings the temperature of the metal must be higher than in casting ingots. The power consumption is increased by the cooling of the furnace between melts. The electrode consumption was also probably increased because the holder clamps wore into the electrode, and the worn part spalled and burned to a point when it got down into the furnace. It will be recalled that this

holder is of different type than that used on the older Héroult furnaces. The difficulty with the lining and roof was increased by the necessarily high temperature of the slag and metal. Also the writer believes that less difficulty in this respect would be experienced if the hearth was made more shallow and the lining thinner. This would prevent the "flaming arc," as the electrodes would be farther from the sides. The arc would be more concentrated beneath the electrode and less heat would be directly radiated to the roof.

PRODUCTS.

As previously stated, the product of this furnace is a soft steel containing on an average 0.14 per cent of carbon, the content varying between 0.10 per cent and 0.20 per cent. The manganese and silicon are varied according to the use to which the steel is to be put. Sulphur and phosphorus are kept below 0.03 per cent. The steel produced is equal in all respects to crucible steel maufactured at the same plant.

GIROD STEEL PLANT, UGINE, FRANCE.

The Girod steel plant at Ugine, France, is the first and largest complete steel plant built that uses electric power entirely for furnace purposes. In 1898 Girod began experimental work on the manufacture of ferro-alloys with a small 20-kilowatt furnace. To-day this manufacture has developed into an industry with an annual production valued at about $3,000,000 and using about 22,000 kw. for electric-furnace purposes during periods of high water in the streams. As previously stated, from the electric ferro-alloy furnace was developed the Girod electric furnace. The steel works were erected in 1909, and because of their uniqueness will be described in detail.

LOCATION.

The plant is situated in a place so remote from supplies of coal and raw material that only a product in which one of the chief requisites to commercial success was cheap power could be manufactured profitably there. Ugine is on the line of the Paris, Lyon & Marseilles Railroad between Annecy and Albertville, about 25 miles from Champbery. The plant is about half a mile from the main line, with which it is connected by an electric tramway for freight haulage, which is operated by the Girod company. The buildings are on the north bank of the River Arly.

POWER SUPPLY.

The total capacity of all the Girod power plants is about 28,000 kw., of which about 6,000 kw. is used at the steel plant. There is a possible development of 30,000 kw. more, a total of 58,000 kw. Not all

of this is primary power, however; that is, power 24 hours a day, 365 days a year. The average cost of power from all sources is about $18.66 per kilowatt-year, or 0.2 cent per kilowatt-hour, based on a flat rate. There are three separate power plants, two on the Arly River and one on the Bonnant River. Some power is leased at times of low water. The power is delivered at the steel works at 45,000 volts and is stepped down to the desired voltage.

DESCRIPTION OF PLANT.

All buildings are of stone and cement construction. The stock house at the rear of the furnace house is 58 by 460 feet. It is used for the receipt and storage of steel scrap and furnace materials of all kinds. There are overhead traveling cranes and lifting magnetic cranes, used to unload carloads of scrap iron and steel.

The furnace and casting house is 70 by 550 feet. In it there are two 10 to 12.5 ton 3-phase Girod furnaces (fig. 30) and three 2.5 to 3 ton single-phase furnaces (fig. 28), which have already been described (pp. 77–78). The three small furnaces are operated with 300 kw. supplied by a 25-cycle current at 60 to 65 volts. The larger ones take 1,000 to 1,200 kw., using a 70 to 75 volt, 25-cycle current. The furnaces are placed on a concrete foundation about 6 feet above the ground floor. The working floor of the furnace is concrete, and the furnaces are set at its edge as in an open-hearth plant. The plant is arranged for a production up to 200 tons a day. One of the smaller furnaces is used entirely for small casting work. The other two produce high-priced alloy steels. The two 10 to 12.5 ton furnaces are used for making carbon steels and the more common alloy steels. In front of the furnace platform are casting pits, one end being used entirely for foundry work. The furnace house is provided with two 2-ton traveling electric cranes and two 12-ton traveling electric cranes.

The rolling-mill building is 90 by 250 feet and contains 1 train of three-high rolls capable of rolling ingots 20 inches in diameter and 400 kg. (880 pounds) in weight to rods 5 inches in diameter. Another three-high mill rolls rods 13 inches in diameter to smaller rods of round, square, and other shapes. The trains are driven by a 600-kw. 3-phase motor. This building also contains 2 producer-gas fired furnaces for preheating ingots.

A large forging shop 70 by 250 feet contains 9 hammers operated by compressed air. The largest hammer weighs 5,000 kg. (11,000 pounds) and the others weigh from 1,000 kg. (2,200 pounds) to 100 kg. (220 pounds). A second forging shop contains one 1,000-ton forging press, 1 forging hammer weighing 10 tons, and 3 stamps with falling hammers, weighing, respectively, 3 tons, 2 tons, and

1 ton. In the first shop rough forgings are made, primarily for automobile machinery, shaftings, gears, tool-steel rods and projectiles.

A tempering shop 50 by 200 feet contains furnaces for tempering, annealing, and hardening large forged or cast-steel pieces. An annealing shop 33 by 67 feet contains several furnaces.

The steel foundry contains a carpenter shop for making patterns, a sand-separating plant, molding machines, molding frames, drying and heating ovens, and a sand-blast jet for cleaning finished castings. The foundry is able to produce 10 tons of steel castings daily, but castings weighing 20 tons have been cast.

The power house at the plant, which is 50 by 115 feet, is equipped with 4 electrically driven compressors and 2 rotary converters.

The warehouse for finished products is 50 by 325 feet, and is made especially large because of the variety of products, and also because it is necessary to carry common shapes over in stock during low-water periods.

In addition to the buildings mentioned there are a furnace-transformer house, a sand-storage house, a drying-oven house, an assembly shop, a machine shop, a storage house for the rolling mills, a pattern warehouse, general stores, a well-equipped physical and chemical laboratory, and an office building.

REFINING PRACTICE.

The refining of cold scrap at the Ugine plant[a] is typical of electric-furnace practice in work of this nature, and is here described in detail. Almost any grade of scrap steel or scrap wrought iron is charged into the Ugine furnaces, the mean proportions of carbon, silicon, manganese, sulphur, and phosphorus in the charge being as follows:

Average proportions of carbon, silicon, manganese, sulphur, and phosphorus in charge.

	Per cent.
Carbon	0.3 to 0.4
Silicon	.1 to .3
Manganese	.6 to .8
Sulphur	.07 to .120
Phosphorus	.07 to .12

The refining operation can be divided into two periods, the oxidation period, in which phosphorus is removed from the metal, and the deoxidation period, with the elimination of sulphur.

After the furnace has been charged with cold scrap, an oxidizing flux of lime and iron ore is added. The proportions vary with the charge and product desired, but a mixture commonly used is 80 kg.

[a] Girod, P., Studies in the electrometallurgy of ferro-alloys and steels: Trans. Faraday Soc., vol. 6, 1911, p. 172; The electric steel furnace in foundry practice: Metall. Chem. Eng., vol. 10, 1912, p. 663; Borchers, W., Electric smelting with the Girod furnace: Trans. Am. Inst. Min. Eng., vol. 41, 1910, p. 120.

(176 pounds) of lime and from 220 to 250 kg. (485 to 550 pounds) of iron ore. Oxidation of the impurities in the metal begins as soon as the scrap becomes pasty, so that by the time the charge is completely melted the carbon, silicon, manganese, and phosphorus contents of the metal are usually reduced to less than 0.1 per cent each. If this is not the case, the slag is withdrawn by tilting the furnace backwards and another flux of the same composition is added. To cleanse the bath of the last traces of the phosphorous-bearing slag and any phosphorus in the metal, lime is added and the resulting slag removed. This is repeated as often as may be necessary. In this first period the temperature is gradually increased, but the oxidation takes place largely at a low temperature.

When the oxidizing and cleansing slag has been removed, the first deoxidation of the bath is effected by adding reducing agents such as ferrosilicon or ferromanganese, but these alloys are added in such a proportion as not to remain in the bath, serving merely as deoxidizing agents. If a high-carbon steel is to be made, recarburization takes place at this point.

The bath is then covered with a flux consisting of about 5 parts lime, 1 part silica sand, 1 part fluorspar, and a little petroleum coke. In this period the furnace must be tightly closed. For proper deoxidation and desulphurization of the bath all iron oxide in the slag must be reduced. This is done with petroleum coke and deoxidizing alloys such as ferrosilicon, silicomanganese, ferrosilico-manganese-aluminum, or even silicon-aluminum. These alloys act energetically and form fluid slags which rise to the surface.

When the slag is completely deoxidized, which is shown by its being white and disintegrating to a powder in air, any desulphurization not completed in the first period is finished by the passage of the sulphur to the slag as calcium sulphide, the chemistry of the process will be described later (p. 124). Carburizing materials are added for finishing the metal, as well as other alloys to bring the steel to the desired composition. If any of the slag from the first period has been left in the furnace, the phosphorus in it will be reduced in the second period, passing into the steel. The average analysis of the final white slag is as follows:

Average composition of final slag.

	Per cent.
CaO	74.85
SiO_2	13.20
FeO	.13
MnO	Traces.
Fe_2O_3	None.
Al_2O_3	1.75
MgO	4.22
P_2O_5	.09
S	1.20

The power consumption recorded at the terminals of the furnace, including melting, refining, and finishing a charge of cold scrap is estimated by Girod at about 850 kilowatt-hours per ton of metal tapped for a 2.5 to 3 ton furnace; at 750 kilowatt-hours per ton of metal tapped for a 10 to 12 ton furnace. The power consumption varies with the charge and product required. A power factor of 0.80 to 0.88 is maintained.

The electrode consumption is stated to be 8 to 9 kg. (17.6 to 19.8 pounds) for a 2.5 to 3 ton furnace, and 8 to 10 kg. (17.6 to 22 pounds) for a 10 to 12 ton furnace. These figures are based on the use of electrodes made at the Girod plant, not designed for continuous feeding, so that this item might be considerably reduced.

Dolomite-and-tar hearths are used. The life of a lining is 90 to 100 heats for a furnace of 10 to 12 tons and about 120 heats for the 2.5 to 3 ton furnace. At the end of that time the side walls are repaired. All burned or oxidized parts of the walls are scraped. About 4 to 6 inches of the bottom is taken up and retamped with a new layer of dolomite. Care is taken to leave the bottom electrodes clear. The roof of the furnace is made of silica brick and stands on an average 50 heats for the 10 to 12 ton furnace and 70 heats for the 2.5 to 3 ton furnace.

The output of metal is 90 to 96 per cent of the charge, depending upon the nature of the metal charged.

A 2.5 to 3 ton furnace requires for operation 1 melter, 1 assistant, and 1 boy; the 10 to 12 ton furnace 1 melter, 2 assistants, and 1 boy. This does not include foremen, cranemen, ladle men, and other laborers.

PRODUCTS OF THE FURNACE.

An idea of the charges, power consumption, and wide variety of products made at Ugine may be obtained from the following record[a] of the operation of this furnace:

[a] Girod, P., op. cit., and Borchers, W., op. cit.

Operation and products of 2.5 to 3 ton Girod furnace.

Quality of steel manufactured.	Desired proportions of—						Time of starting.	Charge.				Time of pouring.	Production in ingots of 350 kg.	Total production.	Incomplete analysis of metal obtained.								Electrical energy consumed per ton.
	C.	Si.	Mn.	Ni.	Cr.	Va.		Scrap.	Turnings.	Ore.	Lime.				C.	Si.	Mn.	S.	P.	Ni.	Cr.	Va.	
								Kg.	*Kg.*	*Kg.*	*Kg.*			*Kg.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Kw.-hrs.*
Tool steel	0.90	0.30	0.30	.....	.....	.....	7 a. m.	1,900	200	100	60	11.10 a. m.	5½	1,725	0.845	0.27	0.31	0.019	0.013	.....	.....	.....	900
Nickel-chrome steel.	.30	.20	.35	2.8	0.6	.....	11.40 a. m.	2,000	200	100	60	6.00 p. m.	6	2,100	.277	.153	.358	.016	.008	2.86	0.51	.....	950
Nickel steel	.15	.20	.35	5.0	.....	.....	12.25 a. m.	2,300	200	120	70	7.00 a. m.	6½	2,275	.156	.194	.390	.012	.012	5.18	.....	.....	950
Nickel-chrome-vanadium steel.	.30	.15	.45	3.0	1.0	0.30	7.30 a. m.	2,500	200	120	70	2.15 p. m.	7	2,450	.297	.120	.473	.010	.005	3.22	1.06	0.23	900
Steel for shell plugs	.30	.20	.40	.....	.....	.....	3.10 p. m.	2,500	200	120	70	10.40 p. m.	6½	2,370	.312	.176	.420	.006	.013	.....	.....	.....	900
Extra soft steel	.10	.20	.30	.....	.....	.....	11.00 p. m.	2,800	300	200	70	7.00 a. m.	(*a*)	2,775	.097	.190	.317	.014	.012	.....	.....	.....	925

a 7 castings.

FOUNDRY OF MÖNKEMÖLLER & CO., BONN, GERMANY.

At the foundry of Mönkemöller & Co., Bonn, Germany, three Stassano furnaces (figs. 19, 20) are producing low-carbon steel for high-grade steel castings and one additional Stassano furnace of 2-ton capacity is being built. The Stassano furnace in small units is better adapted to this use than to any other.

DESCRIPTION OF PLANT.

There are two 1-ton furnaces, each requiring 200 kw., supplied by a current at 110 volts with a frequency of 25 cycles, and one 2-ton furnace taking 400 kw. at 120 volts; the frequency is 25 cycles. The two small furnaces are of essentially the same construction as shown in figures 19 and 20, having the rotating mechanism. The larger one differs only in that it is set on rollers for tilting. The details of construction are as given in the description of the Stassano furnace.

FOUNDRY PRACTICE.

Scrap steel of various sizes and grades is used as a raw material. This is charged to a point directly beneath the three arcs. The desulphurizing and dephosphorizing operations are in general about the same as at other plants.

It takes about three to four hours to melt the charge and from four to five hours for complete melting and refining. The loss of heat in the water cooling of the electrodes is about 14 per cent. The electrodes consumption is very high, owing to failure to use up stumps. The repair cost is also high. The magnesite brick roof lasts from 80 to 100 heats; the whole furnace must then be relined. The power consumption is about 800 to 1,000 kilowatt-hours per ton. A power factor of from 0.85 to 0.90 is maintained. Power is supplied by a public-service corporation at a cost of about 1 cent per kilowatt-hour. The electrodes are regulated hydraulically, but not automatically, as a man watches the meter for changes all the time.

The metal is poured from the furnace into a 1 or 2 ton ladle, from which it is poured into small shanks for casting. All three furnaces are on the main casting floor of the foundry. Most of the castings made are of small size. The steel has an average content of 0.8 to 0.18 per cent carbon, 0.03 per cent sulphur, and 0.06 per cent phosphorus. Occasionally tool steels, with or without the addition of alloys, are made. The process is said to be much cheaper than the crucible method and gives a better product.

SHEFFIELD ANNEALING WORKS, SHEFFIELD, ENGLAND.

At the Sheffield Annealing Works, Sheffield, England, a 2 to 2½ ton Grönwall furnace is in use for the production of steel from scrap.

The chief product is alloy steel, which is cast into ingots and further treated at another works. The scrap steel used consists for the greater part of small pieces of tool scrap and other alloy scrap.

DESCRIPTION OF PLANT.

The furnace (figs. 24, 25) has been described. At some future time the cables to the electrodes are to be replaced by bus bars. The furnace foundation is set level with the ground, with a pit in front for teeming. The building is not well arranged for electric furnace work; the ventilation is very poor. Two annealing furnaces are adjacent to the casting pit.

The two-phase current is received from the main line of the municipal power system at 2,000 volts, with a frequency of 25 cycles, the cost being about 1 cent per kilowatt-hour. An integrating wattmeter, from which the power is charged to the furnace and from which power figures are taken, is placed on the primary side of the circuit where the line enters the building. The meter is followed in the circuit by choke coils to the two transformers. These transformers can be adjusted to give 50, 60, 70, or 80 volts on the secondary circuit. The usual voltage is 75 volts. The neutral point is connected to the bottom of the furnace and a phase is connected to each electrode. The secondary switchboard has a power-factor meter, which can be thrown into any leg, as well as a voltmeter, with the same adjustment. On the switchboard there is also one ammeter set in the neutral. On the back of the furnace there are two ammeters, one to each phase, by means of which two men regulate the electrodes.

PLANT PRACTICE.

The furnace is operated only 12 hours a day, so that more than two charges are never run in one day; also, in the wintertime, it is necessary not to draw too heavily on the power system late in the afternoon when the load on the city lines begins to get heavy. Such operation, of course, is not economical with respect to power consumption, as the furnace cools overnight. Constant change of temperature in the furnace also increases the wear on the lining and the electrodes.

At the time the furnace was inspected it was charged with steel scrap containing 0.40 per cent carbon, 1 to 2 per cent tungsten, and about 1 to 2 per cent chromium. The object of the run was melting with some refining and the casting of ingots. The charge melted in 2 hours and 15 minutes, during which period 560 kilowatt-hours per ton were expended, and was poured in 4 hours and 5 minutes, with a total expenditure of energy of 902 kilowatt-hours per ton. Thus 62 per cent of the energy consumed was used for melting. During the melting stage the amperage was held at 4,000 amperes, or about

500 kw. was on the furnace, while during the refining period this was reduced to 2,500 to 3,000 amperes and about 200 to 300 kw. Often only about 1,500 amperes are on toward the end, but at the very end the temperature is raised to keep the slag fluid. This is said to protect the roof, which seems to be more attacked during the last half hour than any other time. During the melting period a mixture composed of 100 pounds of lime, 30 pounds fluorspar, and 20 pounds of silica was added in two parts. None was added after the melting period, but as soon as the alloy was hot anthracite coal dust was used for deoxidation. In 1½ hours the slag became white. At the end of 3 hours and 50 minutes samples of metal and slag were taken. The slag was white. Analysis of the steel showed a carbon content of 0.39 per cent. While the analysis was being made more coal was shoveled in to prevent reoxidation of the slag. Another sample of slag was taken and the charge poured into a 2.5-ton ladle that had been preheated. From the ladle it was teemed by top-casting into ingots. About 20 pounds of 50 per cent ferrosilicon were placed in the spout before pouring, and 20 to 30 pounds of 80 per cent ferromanganese were added to the charge in the furnace. The weight of metal tapped was 2.1 tons. Some aluminum was added to the tops of the ingot molds.

The electrode consumption is about 20 pounds per ton, at a cost of about 4 cents a pound. Under the poor operating conditions the silica roof lasts about 20 heats. The conducting bottom had been in 60 heats at the time of inspection and was still in good condition.

The furnace force consists of a superintendent, a chemist, a melter, two assistant melters, a ladle man, and one general laborer.

PLANT OF CRUCIBLE STEEL CASTING CO., LANSDOWNE, PA.

At Lansdowne, Pa., the Crucible[a] Steel Casting Co. has recently built the first Röchling-Rodenhauser steel furnace to be used commercially in the United States. It is used for producing steel for castings from cold scrap.

The current used is single-phase and the furnace is of 2 tons capacity, similar to the furnace shown in figure 27. It is situated 200 feet from the generator, on an operating platform. The lining is a mixture of magnesite and tar and is tamped by hand. The roof is magnesite brick. A motor-driven blower supplies the air for cooling the transformer coils for the furnace.

A 300-kilowatt generator supplies a 25-cycle single-phase current at 480 volts, and is directly driven by a 2-cylinder oil engine. The power cost is estimated at 0.7 cents per kilowatt-hour.

[a] Von Bauer, C. H., The Röchling-Rodenhauser furnace of the Crucible Steel Casting Co., Lansdowne, Pa.: Iron Trade Rev., vol. 53, 1913, p. 153.

The furnace is charged while hot with cold scrap, some molten steel from the previous run being left in the furnace. At present it is operated only 12 hours a day, making two melts in that period. The general refining process is the same as in other electric steel furnaces. The length of run varies from four to six hours and the power consumption from 800 to 900 kilowatt-hours per ton. Steel has been produced of the following composition—0.30 per cent carbon, 0.30 per cent silicon, 0.49 per cent manganese, 0.03 per cent phosphorus, and 0.03 per cent sulphur.

SUPERREFINING OF MOLTEN STEEL IN THE ELECTRIC FURNACE.

PLANT OF ILLINOIS STEEL CO., SOUTH CHICAGO, ILL.

The 15-ton 3-phase Héroult[a] furnace at the South Chicago works of the Illinois Steel Co. until recently was one of the two largest electric furnaces in operation for the manufacture of steel. A wide variety of products was made in an attempt to learn what could be done with the electric furnace. The furnace was primarily intended for the refining of molten Bessemer steel, but some cold scrap has been treated.

DESCRIPTION OF PLANT.

The Héroult furnace at South Chicago has already been described. The steel operating platform of the furnace is about 9 feet above the ground level. Around the furnace on this platform, at convenient points, bins are placed for the materials used in furnace operation. The front part of the furnace platform opens to allow a ladle to be hung in position when steel is poured.

Power for the furnace is supplied by the central plant of the works. Dynamos are driven by reciprocating gas engines, reciprocating steam engines, and high-pressure and low-pressure turbines. Blast-furnace gas is used in the gas engines and under boilers. The power cost is about 0.5 cent per kilowatt-hour. The 25-cycle 3-phase current is stepped down at the furnace from 2,200 volts by three 750-kilowatt transformers. These transformers are arranged so that the number of turns in the primary may be altered to give a secondary voltage of 80, 90, 100, or 110 volts. The usual voltage is 90 volts.

The pouring platform, 30 feet long, enables eight molds to be placed in position for pouring. The furnace is served by a 50-ton crane.

PLANT PRACTICE.

Bessemer pig iron is full blown, in a 15-ton Bessemer converter, in 8 to 12 minutes. The analysis of the Bessemer metal shows 0.05 to

[a] Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 205; The South Chicago electric furnace plant of the U. S. Steel Corp.: Met. and Chem. Eng., vol. 8, 1910, p. 179.

0.10 per cent carbon, 0.005 to 0.015 per cent silicon, 0.05 to 0.10 per cent manganese, 0.035 to 0.07 per cent sulphur, and about 0.095 per cent phosphorus. It is then poured directly from the converter to a transfer ladle and drawn to the electric furnace building, about one-fourth of a mile away. As a precaution against the possible formation of a skull in the ladle, the Bessemer charge is blown hotter than in ordinary Bessemer practice.

When the ladle is received at the electric furnace it is lifted by a crane and tilted; the silicious slag is removed by hand rabbling. The metal is then poured into the electric furnace through a spout. The cleaning of the slag and the charging take 5 to 10 minutes. As the metal pours into the furnace the helper shovels iron oxide and lime through the furnace doors to produce a basic oxidizing slag for the removal of phosphorus. In about 30 minutes the furnace is tilted forward and the slag removed in 5 to 10 minutes by hand rabbling. The quantities of carbon, silicon, manganese, and phosphorus have been reduced to a low point in this period.

In the second or deoxidizing stage which now begins sulphur is eliminated as much as possible. The recarburizing agent is added, followed by lime and fluorspar to keep the mass fluid. In about 15 minutes this second slag is fluid, and finely divided coke dust, the deoxidizer, is thrown on top of the slag, forming a reducing atmosphere, as shown by the formation of calcium carbide beneath the electrodes. Thereafter there is a dead melting in a reducing atmosphere. The slag is basic and fluid and contains considerable calcium carbide. It will be seen that the refining practice with molten metal is quite similar to that with cold scrap.

Tests are taken to show the condition of the steel. A small cylindrical test piece is poured and then flattened under a steam hammer at the furnace. If the forged sample appears to be satisfactory, the bath is tapped. The furnace electrodes are raised, and the charge is poured into a ladle, from which it is teemed into ingots.

The quantity of material used in a typical charge is given in the following charge sheet:

Charge sheet of 15-ton Héroult furnace, South Chicago, Ill.

Materials used.	Pounds.	Pounds per tons of 2,240 pounds.
Bessemer blown metal	30,000	.....
Scale	700	52.3
Ferromanganese, 80 per cent pure	200	14.9
Ferrosilicon, 10 per cent pure	60	4.5
Ferrosilicon, 50 per cent pure	80	6.0
Recarbonizing agent	130	9.7
Fluorspar	400	29.9
Coke dust	200	14.9
Lime, first slag	600	45.0
Lime, second slag	600	45.0
Dolomite lining	400	29.9
Magnesite lining	25	1.8

The time consumed in the different operations is divided as follows:

	a. m.
Previous heat tapped	7.00
Metal ordered for	7.15
Metal received	7.15
Fettling begun	7.17
Current on	7.27
Removal of slag begun	8.00
Removal of slag finished	8.11
Tapped	8.48

Time of heat, 1 hour 21 minutes.

The temperature of the Bessemer metal poured into the electric furnace is almost 1,500° C., whereas that of the steel poured from the furnace is about 1,495° C. At the close of a run the furnace temperature drops from about 1,500° C. to 1,300° C. during the 15 minutes necessary to patch the lining for the next run.

On an average 12 heats are made daily. During a series of experiments in which both dephosphorization and desulphurization took place, the average power consumption was 198 kilowatt-hours per long ton. When desulphurization alone was carried on, the average power consumption was 105.9 kilowatt-hours per long ton. The power factor of the furnace varies from 0.80 to 0.90.

The hearth is fettled after each run with dolomite, from 10 to 30 pounds of dolomite being used per ton of steel. The hearth proper consists of 4 parts dead-burned, ground magnesite mixed with 1 part basic open-hearth slag. To this mixture tar enough is added to make the mass plastic for tamping. After being tamped, the furnace is heated with a wood fire for 48 hours; it is then filled with coke, the current turned on, and the bottom fluxed into place. This bottom and the side walls last a long time. A silica-brick roof withstands about a week's use and is repaired or replaced during the regular weekly shutdown. The roof has then been subjected to about 72 heats, although some roofs have stood 129 heats. A new roof is always kept in reserve.

Extensive tests have been made with various kinds of electrodes. The average consumption of either amorphous carbon or graphite electrodes is given as about 6 pounds per ton of steel. As in cold-scrap refining processes, the electrodes are regulated by hand during the first few minutes of a run.

PRODUCTS.

This furnace has produced a greater variety of steel than any electric furnace in operation, the product including high-grade alloy steel, high-grade carbon steel, and ordinary carbon steel. From the ordinary carbon steel produced, rails of a dozen different sections,

billets of all sizes and grades, plates of all sizes and grades, structural shapes, castings, both small and large, and high and low carbon, and forgings of all sizes have been manufactured. Among the alloy steels made are nickel, nickel-chrome, chrome, manganese, and silicon steels. Some steel has been made from cold scrap.

The main characteristics of the steels produced in the furnace are a comparative freedom from oxidation and from segregation and a higher tensile strength and slightly higher ductility than that of open-hearth or Bessemer steel of the same chemical composition up to 0.4 per cent carbon, where the difference becomes less apparent.

PLANT OF AMERICAN STEEL & WIRE CO., WORCESTER, MASS.

A 15-ton, three-phase Héroult furnace similar to the South Chicago furnace was in operation until recently at Worcester, Mass., for the production of steel for wire from basic open-hearth steel. Owing, presumably, to difficulty in drawing the comparatively high-carbon steel into wire the furnace is not operated regularly at present. The furnace is similar in construction to the furnace at South Chicago. A current of 12,000 volts is received and is transformed to about 110 volts or less at the plant, according to the voltage desired for the furnace. There are three 750-kilowatt transformers, and the furnace is connected to the transformers by bus bars. Electrode regulators similar to those at South Chicago are used. The cost of power is 0.6 to 0.8 cents per kilowatt-hour.

The process is desulphurizing only and hence requires less time than at that at South Chicago. The procedure of one run was as is stated below. The charge of molten basic open-hearth steel contained 0.18 per cent carbon, 0.22 per cent manganese, 0.02 per cent silicon, 0.02 per cent phosphorus, and 0.027 per cent sulphur. As the 12 to 15 ton charge was poured into the electric furnace about 280 pounds of electrode dust were added. Charging took about 5 minutes. When the furnace was charged a mixture made of 700 pounds lime, 350 pounds fluorspar, and 50 pounds of coke was added. In about 30 or 35 minutes 120 pounds of fluorspar were rabbled in. This was followed in 10 minutes by 30 pounds more of fluorspar. About an hour after the charging was completed a sample was taken. One of these contained 0.50 per cent carbon, 0.12 per cent manganese, and 0.019 per cent sulphur. Ferromanganese and pig iron were then added. About 10 minutes after the first sample had been collected another was taken, which contained 0.45 per cent carbon, 0.45 per cent manganese, and 0.02 per cent sulphur. The slag at this time was tested for calcium carbide, then ferrosilicon, sand, and lime were added. The current was then turned off and the charge poured into a ladle and cast into ingots. The finished product contained

0.54 per cent carbon, 0.47 per cent manganese, 0.157 per cent silicon, 0.022 per cent of phosphorus, and 0.015 per cent of sulphur.

The average current strength for a run was 12,310 amperes; the power factor was 0.696; the voltage was 76.2, 97.7, and 73.3 volts; and the power 396, 451, and 404 kw. to the respective phases. The sum of the kilowatts on the three phases was 1,253 kw. and the average length of a run was 1.41 hours, making the total power consumption 1,746 kilowatt-hours. The average power consumption per ton of metal poured is about 152 kilowatt-hours.

The temperature of the steel when charged is about 1,454° C. The temperature of the slag is 1,524° C. The steel is tapped at a lower temperature than it is charged.

The total losses in bus bars, electrodes, and transformers is 6.53 per cent of the current measured at the switchboard. About 2 per cent of this loss is in the transformers. The loss by conduction through the walls is 5.57 per cent of the total power supplied the furnace.

GUTEHOFFNUNGSHÜTTE WORKS, OBERHAUSEN, GERMANY.

An extensive test[a] has been conducted at Oberhausen, Germany, on the Girod electric furnace (fig. 32) for the refining of steel from molten basic open-hearth metal.

DESCRIPTION OF PLANT.

The Girod furnace installed here is from 2.5 to 3 ton capacity (fig. 22) and is similar to the usual type of Girod furnace.

The single-phase current is supplied by a motor-generator set consisting of a 3,000-volt, 95-ampere, 575-kilowatt, three-phase induction motor driving a single-phase 75-volt, 6,700-ampere, 25-cycle, 500-kilowatt generator. The power factor is 0.8. These machines, as well as all conductors carrying high-tension currents, are placed in a room adjoining the furnace. The loss in transforming from three-phase to singe-phase current is about 14 per cent. On the furnace switchboard there are ammeters and voltmeters in the low-tension circuit, with a wattmeter in the primary circuit. The excitation-current regulator for the furnace voltage, the manual and automatic regulator for the electrodes, and the device for tilting the furnace are all controlled from the switchboard.

REFINING PRACTICE.

In continuous operation it is possible at this plant to run 8 heats in 24 hours, a total of 25 tons of steel. The basic open-hearth furnaces

[a] Mueller, A., The manufacture of steel in the Girod electric furnace: Metall. Chem. Eng., vol. 9, 1911, p. 581.

are tapped only every four hours, so that the electric furnace can not be operated continually. The furnace is charged by means of a 23-foot launder extending from the open hearth. During the charging process the current is 4,000 to 6,000 amperes. After the charging

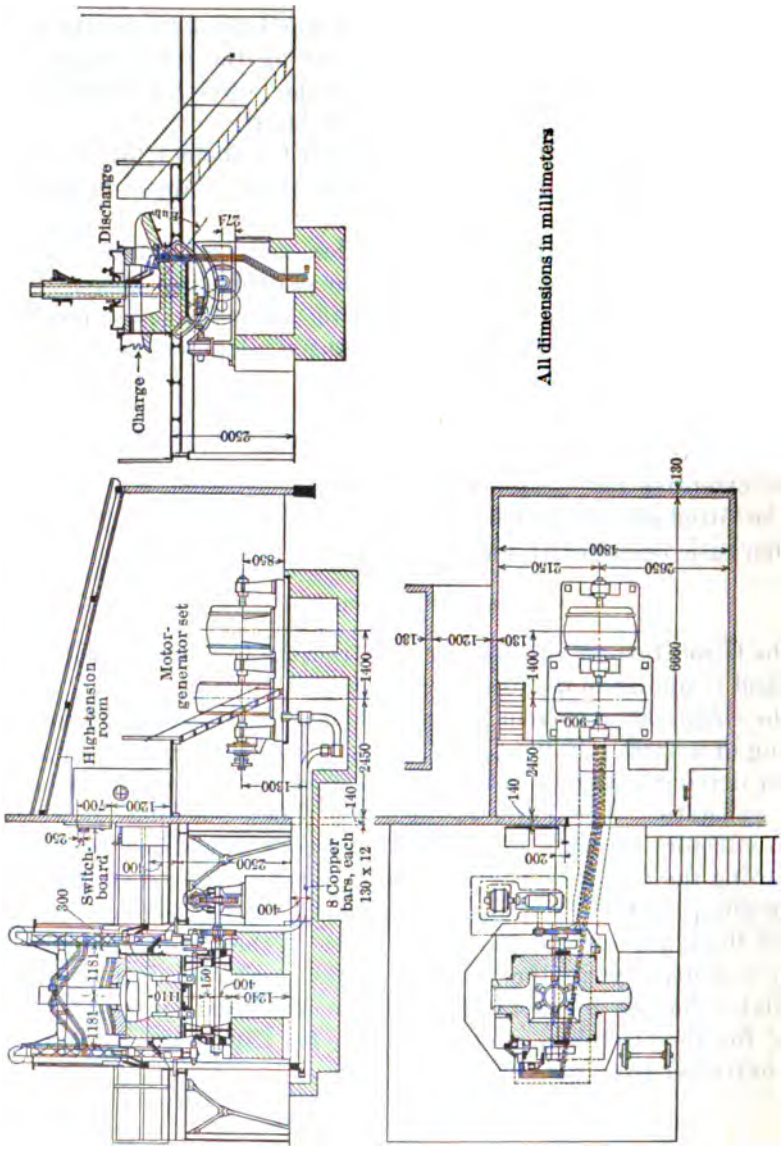

FIGURE 32.—Arrangement of electric-furnace steel plant, Oberhausen, Germany.

launder has been removed from the furnace the flux of ore and lime for oxidizing is shoveled into the furnace. The usual practice is followed for oxidation and deoxidation. The results of tests to determine oxidation and deoxidation of impurities is given below.

Figure 33 shows diagrammatically the chemical reactions during a heat in the Girod furnace and illustrates the changes of the carbon, manganese, phosphorus, sulphur, and silicon contents.

The results of the analyses of 19 samples are here plotted. Sample 1 is that of the charge, sample 2 was taken after addition of 15 kg. (33 pounds) of ore, sample 3 after addition of 50 kg. (110 pounds) of ore. Samples 4 and 5 were also taken during the oxidation period, sample 5 being taken just before removal of the oxidation slag. Samples 6 to 19 were taken during the deoxidation period. Sample 6 was taken after the addition of 16 kg. (35 pounds) of Girod carbons and 5 kg. (11 pounds) of ferromanganese; sample 7 after the addition of 12 kg. (26.5 pounds) of ferrosilicon (50 per cent Si) and 40 kg.

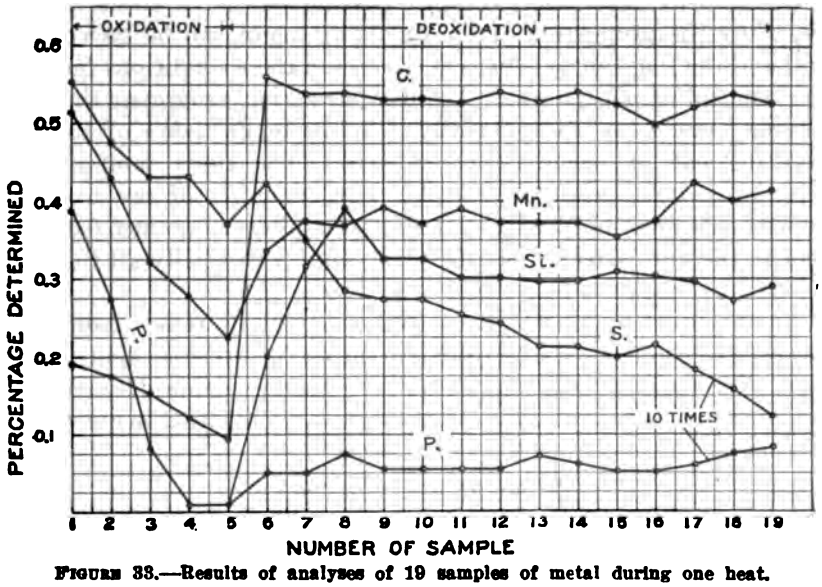

FIGURE 33.—Results of analyses of 19 samples of metal during one heat.

(88 pounds) of refining slag; sample 8 after the addition of 4 kg. (8.8 pounds) of ferrosilicon; sample 9 after the addition of 20 kg. (44 pounds) of refining slag; sample 10 after the addition of 3 kg. (6.6 pounds) of ferrosilicon; sample 11 after addition of 2 kg. (4.4 pounds) of powdered petroleum coke; sample 12 after the addition of 1 kg. (2.2 pounds) of ferromanganese (80 per cent Mn); sample 13 after the addition of 8 kg. (17.6 pounds) of lime; nothing was added when samples 14 and 15 were taken; sample 16 was taken after the addition of 10 kg. (22 pounds) of lime and fluorspar and 2 kg. (4.4 pounds) of powdered petroleum coke; and sample 17 after addition of 2 kg. (4.4 pounds) of ferromanganese. Nothing was added after that, sample 19 being the last.

The oxidation period must be preceded in many cases by a melting period of a length corresponding to the interval between charges. A longer interval causes the steel to solidify in parts. During this time ore is added gradually to the bath in the furnace.

The carbon oxidizes slowly in spite of the high temperature in the furnace. The curves shown in figure 33 are based on data obtained from a heat in which 0.10 per cent carbon was oxidized in 55 minutes, the amount oxidized being equivalent to 53 per cent of the original carbon content. Although the manganese was present in a more concentrated state than the carbon, only 58 per cent of the original manganese content was oxidized in the same time. The amount oxidized is equal to a decrease in manganese from 0.52 to 0.22 per cent.

Phosphorus can be more completely removed than the other impurities since the proportion of the oxidizing agent can be governed at will and the slag can be kept very basic, and these conditions facilitate the removal of phosphorus. As decarburization progresses the phosphorus content decreases steadily, the low temperature of the bath favoring the oxidation of phosphorus. The third sample taken (fig. 33), after a charge of 50 kg. (110 pounds) of ore had been added, contained only 0.008 per cent phosphorus, against 0.039 per cent in the first, a reduction of 80 per cent. Practically all the phosphorus is removed before slagging proceeds.

As shown in figure 33, the sulphur content decreases from 0.055 per cent to 0.037 per cent during oxidation, the decrease being 33 per cent. Wüst claims that slag containing ferrous oxide dissolves iron sulphide to a certain extent. In stronger concentration it reacts with ferrous oxide, according to the equation:

$$FeS+2FeO=3Fe+SO_2.$$

The sulphur dioxide escapes and can be clearly recognized. But a part of the sulphur may still remain dissolved in the slag together with free ferrous oxide, as is shown by the sulphur in the oxidation slags. The average decrease in sulphur content during the oxidation period is 26 per cent.

The oxidation slag is drawn off, and the last trace of phosphorus is removed from the bath by mixing lime with it. This method avoids rephosphorization of the bath by the carbon and silicon additions in the following period. One slag is sufficient to remove all the phosphorus except traces.

The deoxidizing slag is produced by adding to the bath about 20 kg. (44 pounds) of lime, 3 kg. (6.6 pounds) of sand, and a like quantity of fluorspar, and also 1 kg. (2.2 pounds) of ferrosilicon (50 per cent silicon) per ton of steel. This slag dissolves the ferrous oxide that has been forming on the unprotected surface of the metal and thereby becomes black.

The reduction of the ferrous oxide in the slag is indispensable for the deoxidation and desulphurization of the bath. For this deoxidation the silicon from the added ferrosilicon is not sufficient, and 1 to 2 kg. (2 to 4 pounds) of powdered petroleum coke are added to insure complete deoxidation. The slag obtained disintegrates in the air quickly to a white powder, indicating that the ferrous oxide has been reduced to traces.

With high-carbon products, petroleum coke or waste pieces of carbon electrode are added in proper proportion to the bath as required for purposes of carburization prior to the formation of the deoxidation slag. As soon as the slag is readily fluid and free from oxide, ferromanganese, ferrosilicon, and other alloys are added.

The addition of these alloys which are often impure may increase the phosphorus and sulphur content considerably. Thus, in figure 33 sample 6 shows that the phosphorus has increased from traces to 0.08 per cent, owing to the addition of ferromanganese with a phosphorus content of 0.4 per cent. The chemical composition of the slag at different periods of the melt follows:

Composition of slags.

Constituent.	Oxidizing slag.	Deoxidizing slag.		Finishing slag.
		After carburization.	After main charge.	
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
CaO	41.69	59.50	69.45	75.85
SiO_2	9.58	26.78	17.90	13.20
FeO	28.36	1.02	.27	.13
MnO	.99	1.44	.52	Trace.
S	.62	.39	.90	1.22
Fe_2O_3	6.43	.23		
Al_2O_3	1.73	1.67	1.27	1.73
MgO	6.99	5.54	5.78	4.22
P_2O_5	1.47	.07	.12	.09
Oxide of bases	.20	.20	.22	.22
Oxide of acids	9	15	10	8
Appearance of slag	Black and streaky.	Grayish brown sand.	White sand.	White powder.

The sulphur is removed in three ways: During oxidation, through the formation of sulphur dioxide by the action of slag rich in ferrous oxide; in the deoxidation period, through the action of the white slag, free from ferrous oxide; and through the union of silicon with sulphur, forming silicon sulphide.

The quiet melting of a cold scrap-steel charge has been recognized as a great advantage of the Girod furnace. Figure 34 shows the fluctuations of the kilowatt consumption as traced by a recording wattmeter, starting with a liquid charge. The regulation of voltage and current is almost always automatic; hand regulation, with the auto-

matic regulator disconnected, is used only during the beginning of a run. In figure 34 the letters represent the time of the operations as given: A, tapping; B, C, and D, additions of flux; E, carburization; F, slagging; G, slag readily fluid; H, bath boiling; I, additions of ore; J, slag not fluid; K, charging.

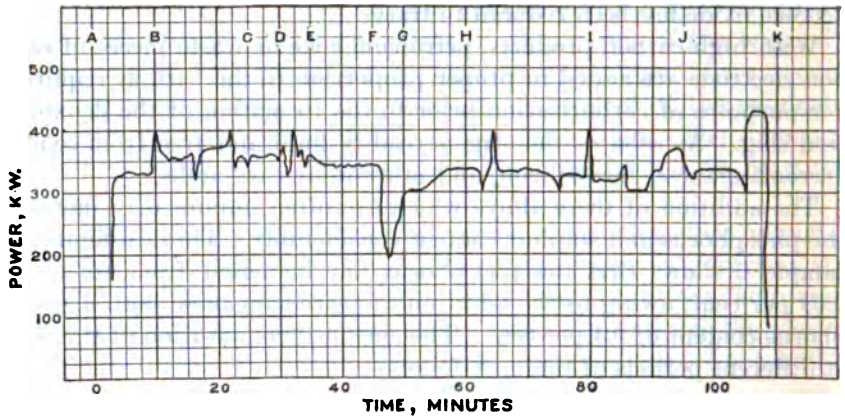

FIGURE 34.—Power fluctuations of 2.5-ton, single-phase, Girod steel furnace during a heat.

The size of the furnace, the duration of the intervals between successive heats, the cooling of the furnace during these intervals, the condition of the charge, the quality of steel desired, and other factors all affect the energy consumption decidedly.

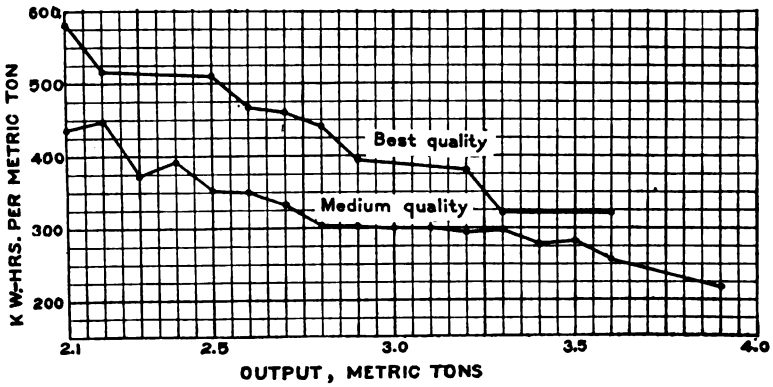

FIGURE 35.—Specific-energy consumption as function of weight of charge, Girod furnace.

Figure 35 shows the influence of the weight of charge on the specific-energy consumption per ton. The results are averages taken from 500 successive heats, including some of alloy steels. The energy consumption depends a great deal on the weight of the charge and decreases in large furnaces.

The loss of heat through radiation and conduction and the fluctuations of temperature decrease with increasing size of furnaces, the temperature being much more easily regulated in large furnaces than in small ones.

The minimum economical weight of charge is about 2,800 kg. (fig. 35), the specific-energy consumption not changing materially with increasing weight up to 3,300 kg.

The specific-energy consumption depends still more on the temperature of the furnace, and this depends greatly on the duration of the intervals between charges. The fall of temperature after tapping is rapid compared with that of an open-hearth furnace, the temperature of which decreases to only 1,000° C. in 25 hours and decreases after-

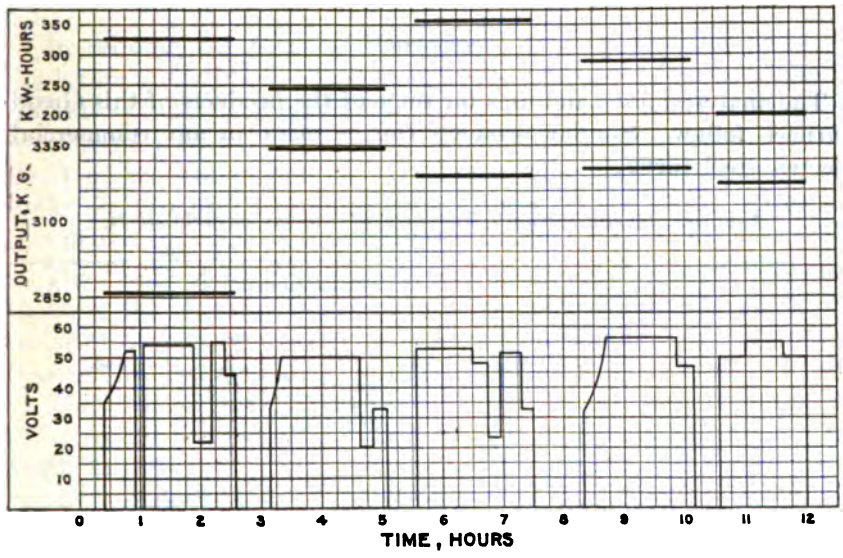

FIGURE 36.—Voltage, output, and power consumption for 2.5-ton, single-phase, Girod steel furnace.

wards at the rate of 20° C. per hour. Under such conditions it is possible to get a high thermal efficiency with the Girod furnace only by charging it, at the latest, 20 minutes after the last tapping. The temperature of the interior of the furnace is then still about 1,300° C. The energy consumption per ton is then 188 kilowatt-hours for medium-grade steels and 270 kilowatt-hours for high-grade steels (fig. 36).

Under favorable conditions it was possible in a Girod 3-ton furnace to refine 3,654 kg. (8,039 pounds) of steel with an energy consumption of 158 kilowatt-hours per ton, including refining by oxidation, deoxidation, and desulphurization. Large Girod furnaces give more favorable results, as they can be charged immediately after tapping, while they are at a temperature of 200° C. higher.

The silica brick roof of the furnace lasts for 60 to 70 heats. The walls are attacked most at the slag line. This is fettled between charges. After 120 heats it is necessary to reline the walls. The hearth bottom is also repaired then, as by that time it has sunk about 2 inches in the center. The hearth does not have to be pulled out when it is repaired.

The heat loss due to cooling of the bottom electrodes is 1.01 per cent of the energy consumption, and in producing 3.5 tons of steel the cooling of these electrodes consumes about 2.9 kilowatt-hours per ton of steel poured. The heat lost in cooling the silica roof around the carbon electrode is 3.65 per cent of the total energy supplied. With a production of 3.5 tons, 10.5 kilowatt-hours per ton would be lost in cooling the upper electrode.

PRODUCTS.

The analyses, tests, and uses for some of the products of this Girod furnace follow. Similar products can be made in any commercial electric steel furnace.

Analyses, tensile strength, and uses of various electric steels.

Va.	W.	Cr.	Ni.	C.	Mn.	P.	S.	Si.	Tensile strength.			Use.
									Weight per square millimeter.	Elongation on 200-millimeter bar.	Reduction of cross section.	
P.ct.	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*P.ct.*	*Kg.*	*P.ct.*	*P.ct.*	
.....	.....	.....	.....	0.13	0.54	Tr.	0.014	0.16	41.2	30	65.6	Angle.
.....	.....	.....	.....	.13	.70	0.018	.020	.08	43	31	62.3	Do.
.....	.....	.....	.....	.19	.56	Tr.	.010	.13	47	31	55	Do.
.....	.....	.....	.....	.22	.90	.010	.017	.25	58	27.5	52.8	Iron.
.....	.....	.....	.....	.25	.88	Tr.	.017	.26	60	23.5	49.1	I-beam.
.....	.....	.....	.....	.33	.74	.010	.010	.20	65	19	42	Engine and machine parts.
.....	.....	.....	.....	.64	.62	.010	Tr.	.47	86	13.2	23.5	Square blocks.
.....	.....	.....	.....	.53	.68	.008	.010	.37	86.9	12.8	30.3	Flat steel bars.
.....	.....	.....	.....	.57	.70	Tr.	.008	.40	89.4	10.7	21.4	Do.
.....	.....	.....	.....	.72	.42	.020	.010	.34	97.1	6	23.8	Dies.
.....	.....	.....	.....	.82	.48	.006	.010	.25	99.5	5.8	20.1	Punches.
.....	.....	.....	.....	.70	.72	.010	.010	.36	83.8	11.5	19.8	Tramway car-wheel tires.
.....	.....	.....	.....	.47	.78	.010	.012	.24	71.5	17.5	32.9	Railway car-wheel tires.
.....	.....	.....	.....	.36	.78	.013	.010	.17	64	19	40.7	Axles.
.....	.....	.....	.....	.48	.78	.007	.019	.28	74.4	13.3	24.9	Locomotive wheel tires.
.....	.....	.....	.....	.52	.78	.015	.018	.19	78.7	11.1	20.5	Do.
.....	.....	.....	.....	.48	.84	.007	.018	.23	72	17.7	44	Shafting.
.....	.....	.....	.....	.....	.....	.....	.....	.....	72	16.5	45.2	Do.
.....	.....	.....	1.89	.26	.78	Tr.	.012	.12	61.8	23.5	48.6	Spindles.
.....	.....	.....	1.98	.20	.72	.012	.018	.13	54	22	58.6	Do.
.....	.....	1.04	.....	1.62	.58	Tr.	.012	.16	.....	.....	.....	Knives, shears, chisels.
.....	1.50	.92	.....	.49	.82	.012	Tr.	.32	79.7	15	40.4	Tools.
0.19	.....	.85	.....	.46	.60	Tr.	.012	.22	79.6	10.2	32.9	Pieces for forging.
.....	.....	.....	.....	.....	.....	.....	.....	.....	62.1	33	59.5	Quenched at 700° C. in oil.
.....	.....	.....	.....	.....	.....	.....	.....	.....	59.6	36	63.8	Quenched at 700° C. in water.
.....	.....	.....	.....	.68	.32	.010	.010	.23	93.3	9	18.8	Rolls for making grooved rail.

WORKS AT VOLKLINGEN, GERMANY.

Two Röchling-Rodenhauser furnaces at Volklingen, Germany, are in operation for the refining of molten basic Bessemer steel, with the production of steel varying in grade from rail steel to high-priced tool steel. At this plant the Röchling-Rodenhauser furnace was developed from the original Kjellin design.

One of the furnaces is of 8 to 10 tons capacity (fig. 27), taking 600 kw., supplied by a 25-cycle single-phase current at a voltage of 4,000 to 5,000 volts. The 2 to 3 ton three-phase furnace requires 200 to 250 kw. at a pressure of 400 volts on a 50-cycle circuit. The details of these furnaces have been previously described.

METHOD OF RELINING FURNACES.

In relining the furnaces the furnace castings are first protected by layers of fire brick, which serve as the principal protection against radiation losses and rarely require replacement. On the fire brick a layer of a mixture of calcined dolomite and tar is rammed to the level of the hearth bottom. Cast iron templates are then set and dolomite mixture is rammed behind them on the sides. After the templates are removed, the walls are finished with dolomite brick.

The lining is dried by placing cast-iron rings in the channels and heating the rings by the passage of current. When the tar in the mixture is burned out the roof is put on and molten pig iron is charged to complete the sintering of the hearth.

The time required for relining a 3-ton furnace is as follows: Removing old lining and ramming in new, 24 hours; burning out tar with hot rings, 6 hours; completing the sintering of the lining, 6 hours—total, 36 hours. During burning and sintering about 230 kilowatt-hours are consumed. Such a lining stands on the average 55 charges, or 360 to 500 tons of hot metal, according to the steel produced.

REFINING PRACTICE.

The molten metal charged into the furnaces at Volklingen from the basic Bessemer converters contains 0.1 per cent carbon, 0.5 per cent manganese, 0.08 per cent sulphur, and 0.08 per cent phosphorus. At times molten basic open-hearth steel that has been refined, containing about 1.22 per cent carbon, 0.38 per cent manganese, and 0.209 per cent silicon, is put in the electric furnace to allow removal of gases, to adjust the carbon content and to add alloys.

Dephosphorizing and desulphurizing are carried on as in the arc furnace, but not at so high a temperature, although the temperature is sufficient for the purpose, and is more evenly distributed through the charge. Also most of the desulphurizing is caused by the action of ferrosilicon and lime on ferrous sulphide, as the temperature is not

high enough to permit the formation of calcium carbide. Hence more ferrosilicon is charged in the induction furnace than in the arc furnace.

The furnace is started cold, rings of steel are placed in the channels, and molten metal is poured on them. The voltage of the secondary current, through the pole plates, is higher than usual at the start in order to warm the hearth more quickly and make it conducting. Formerly it was customary to pass current through the furnace on Sunday to keep it hot, but now the doors are closed and the current is shut off entirely. After the furnace has stood 30 hours part of the current is turned on and the amount is gradually increased until in 4 to 6 hours the furnace is at working temperature. After each run the furnace is practically emptied. In using cold scrap about two-thirds of the charge is molten steel and the rest is cold scrap.

With hot metal the length of a heat varies from 1 hour 15 minutes to 3 hours. With a cold charge the time required is 3 to 6 hours. To produce a rail steel containing 0.5 per cent carbon, 0.8 per cent manganese, 0.04 per cent sulphur, and 0.05 per cent phosphorus, about 100 to 125 kilowatt-hours per ton are necessary with a hot charge of Bessemer metal. For high-grade steel containing 0.05 per cent carbon, 0.25 per cent manganese, and trace of sulphur and phosphorus, the power consumption is about 300 kilowatt-hours per ton. The power factor averages about 0.60 with both the single and the 3-phase furnaces. With cold scrap the power consumption is claimed to be 580 kilowatt-hours per ton. The power cost at Volklingen is about 1.5 cents per kilowatt-hour.

PRODUCTS.

Steel of many grades has been made at Volklingen in the induction furnace, including high-grade tool, alloy, and rail steels. Some of the mild-steel products and the power consumption are as follows:

Mild-steel products of Röchling-Rodenhauser furnace.

	C.	Si.	Mn.	P.	S.	Time.		Power consumption per ton.	Total energy consumed.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Hours.*	*Minutes.*	*Kw.-hrs.*	*Kw.*
Charged		0.028	0.534	0.037	0.081	1			
Poured		.022	.307	.005	.057		50	260	628
Charged		.034	.478	.060	.077	2			
Poured		.018	.309	.011	.053		25	274	685
Charged	0.128	.032	.590	.064	.085	1			
Poured		.140	.618	.020	.055		45	235	587
Charged	.116	.018	.679	.054	.097	2			
Poured		.122	.695	.028	.098		5	282	705

PLANT OF LA GALLAIS METZ & CO., DOMMELDINGEN, LUXEMBURG.

At Dommeldingen, Luxemburg, is the largest installation of Röchling-Rodenhauser furnaces in operation. There are three single-phase 3½-ton furnaces, similar to the one shown in figure 27, and one three-phase 2.5-ton furnace. The single-phase furnaces take 350 to 400 kw. at 3,500 volts, 50 cycles. The three-phase furnace requires 275 kw. at 500 volts, 50 cycles. The power factor of the single-phase furnace averages about 0.6, and that of the three-phase furnace is sometimes as high as 0.7.

Pig iron is made into steel in a gas-heated 20-ton tilting furnace. When 3.5 tons of the steel is transferred to an electric furnace for further refining it is replaced by 3.5 tons of pig iron. The gas furnace acts as an oxidizing mixer. The steel in the gas furnace has 0.26 per cent carbon, 0.082 per cent phosphorus, and 0.03 per cent sulphur. For ordinary castings the steel is in the electric furnace about 2 hours, for the best machinery castings 3 to 3½ hours.

The three-phase furnace was not in use at time of inspection, as faulty design had resulted in poor operation.

Some of the results obtained in practice at Dommeldingen were as follows:

Results obtained from furnace practice at Dommeldingen.

Sample.	Chromium content.	C, Si, Mn, P, and S content.				
		Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
Pig iron		3.50	0.60	1.20	1.80	0.12
Mixture from gas furnace		1.62		.66	.46	.035
Tappings from gas furnace		.26		.30	.08	.030
Steel from electric furnace		.35				
Castings		.40	.25	.50	.020	.010
Hard fine steel		.85	.23	.26	.011	.009
Chrome steel	0.87	1.17	.19	.16	.014	.009

CHARACTERISTICS OF OPERATION OF ELECTRIC STEEL FURNACES.

DESIGN.

From what has been said it is clear that electric steel furnaces vary widely in design, but they may be divided into two general classes, arc furnaces and induction furnaces. The general features of the operation of the furnaces in a class are similar. Steel of about the same grade can be made in any of the commercial electric furnaces, but from their design some furnaces are better adapted to one purpose than to another. In general the power consumption and efficiencies of the various types are about the same. Hence the choice of the type of furnace for a specific purpose should be determined by the factors of the individual case.

Several general points should be considered in choosing the type of furnace. In comparing the arc and the induction furnace it should be remembered that the arc furnace is better adapted to melting cold scrap than is the induction furnace, because the latter must be charged with molten metal at the start to insure quick and regular heats. The power factor of the induction furnace is lower than that of the arc furnace and is much lower than that of most public power service lines. Hence, an induction furnace that might be a satisfactory load on a private plant carrying no other load, on a public line might lower the power factor of the whole system to an objectionable degree. If an induction furnace is built where power must be obtained from a public service line, the company operating the furnace may be required to put in a motor-generator set at its own expense. In general, the induction furnace makes a more steady demand on the central power plant than the arc furnace, because there is no constant breaking of the arc which may cause fluctuations.

In choosing an arc furnace for refining cold scrap it seems that the load on the power line is more satisfactory, if the furnace is of the conducting-hearth type with several upper electrodes in parallel rather than of the type with two arcs in series with the bath. This is because the arc breaks less frequently in the former than in the latter. For continuous steady operation on either cold scrap or molten steel, one furnace is probably about as good as the other as regards mechanical features, power consumption, and heat losses, although some consider the water-cooled electrodes of the conducting hearth and broken hearth to be objectionable. An arc furnace in which heating is by radiation of the arc has a higher power consumption than the other types. Electrode consumption and repair cost are also higher.

POWER CONSUMPTION.

The power consumption of furnaces depends so much upon the materials being treated, the process of operation, and the product desired that no satisfactory comparisons can be made. Also, the point in the circuit at which the reading is taken for figuring the basis of power consumption may be so variously selected that one furnace may show a much lower power consumption than the other when it should show about the same. With cold scrap the power consumption of the various arc furnaces varies from 459 to 1,200 kilowatt-hours per ton. Aside from the points mentioned, this wide variety of results may be due to the amount of refining done. Furnaces in which only one refining slag is used take about 600 kilowatt-hours per ton on an average, while if two slags are used 800 to 1,000 kilowatt-hours per ton are necessary. Power consumption per ton of metal is also influenced by the size of the furnace. For example, a 10 to 12 ton Girod

furnace uses about 12 per cent less power per ton than a 2.5 to 3 ton furnace of the same type.

In refining molten steel in the electric furnace, the power consumption has varied from 100 to 300 kilowatt-hours per ton. Rail steel of this class takes about 100 kilowatt-hours per ton when one slag only is necessary and about 200 kilowatt-hours per ton with two slags in a 15-ton furnace. Tool steel is made from molten metal in a 2 to 3 ton furnace with a power consumption of 150 to 300 kilowatt-hours per ton.

POWER FACTOR.

The power factor is the ratio of the number of effective watts in an alternating-current circuit to the number of volts multiplied by the number of amperes. Any self-induction in the circuit causes a difference in phase between the electromotive force or voltage and the current or amperage, and the current will thus lead or lag behind the voltage; that is, the two do not reach their maxima at the same moment. An amount of energy equal to the number of amperes multiplied by the number of volts is surging through the circuit, but, as the electromotive force and current do not reach their maxima together, the product does not represent the effective energy that is usefully consumed at the receiving end.

The generators have to supply the total wattage and the line has to conduct it. Hence to offset a low power factor large generators and thick conductors must be used. For example, a power factor of 0.5, or 50 per cent, means that conductors of double size must be used to get the same number of effective watts as with a power factor of 1, and it also means that generators of double capacity must be used to produce the same useful output. In such a case there would still be objection raised by the power company, for the voltage regulation of the line and generators would be poor, and the low power factor would act as a brake on the circuit, throwing back on it all power not effectively used and causing the generating machinery to heat. If the company using a furnace with a low power factor generates its own power, it must consider the increased capital investment on account of larger generating capacity.

If the ordinary loads of a central power station are considered, an operating power factor above 0.95 will be obtained only when practically all of the load consists of synchronous motors or rotary converters that may be operated with a power factor of about 1. The power factor for a plant having a large proportion of induction motors is about 0.70. Hence in operating several electric furnaces the necessity of keeping the power factor of the single unit as much above 0.80 as possible is evident or the power factor of the whole plant may drop below 0.70.

Without discussing the causes in furnace design that influence the power factor, it may be said that there are three ways of reducing the trouble. One is to reduce the frequency of the alternating current, but, although this should be considered when a new plant is to be erected, in an established works such a change means new electrical machinery of special design and higher cost. A more practical way is to increase the resistance of the furnace, which results in a higher working voltage and a higher power factor. The third method is to decrease the quantity of iron surrounding the brickwork of the furnace as much as possible, as this iron causes the formation of lines of force that tend to lower the power factor. Tie-rods and buck staves may be made of nonmagnetic special steel, or a strip of copper may be inserted to break the continuity of the iron surrounding the circuit.

The effect of frequency has been mentioned. It is for this reason that most electric steel furnaces are operated with a current having a frequency of 25 cycles rather than with the 60-cycle current ordinarily used in commercial practice. Up to the time of the design of the combination induction furnace the great drawback to the induction furnace as originally built was the necessity of using a current with a frequency as low as 5 to 15 cycles in order to obtain a power factor over 0.5. With the combination induction furnace a power factor of 0.6 can be obtained with a frequency of 25 cycles.

The power factor of a furnace decreases if the size is increased and the voltage and frequency remain the same. For instance, the power factor of a 25-ton single-phase Héroult furnace varies from 0.85 to 0.95 with a current frequency of 25 cycles and a voltage of 110 volts, whereas in a 15-ton three-phase furnace the power factor is 0.70 to 0.80 with a current frequency of 25 cycles and a voltage of 90 volts.

The power factor of the combination induction furnace varies from 0.60 in an 8 to 10 ton furnace, single-phase, frequency 25 cycles, to 0.70 in the 3-ton furnace, three-phase, frequency either 25 or 50 cycles. The power factor of the 1.5-ton induction furnace is about 0.5, using a current with a frequency of 15 cycles. The power factor of induction furnaces, owing to the large proportion of iron used, decreases rapidly with increase in size, so that it would seem that this might prevent an increase in size over 8 to 12 tons.

In general the arc furnace with the conducting hearth tends to have a slightly lower power factor than the arc furnace with the upper electrodes in series and a nonconducting hearth. The power factor of all designs of commercial electric steel furnaces of the arc type is above 0.70 for the sizes over 5 tons, and above 0.80 for the smaller sizes.

ELECTRODES.

With improved methods of making amorphous-carbon electrodes the electrode consumption of electric steel furnaces has been considerably reduced. The manufacture of electrodes with threads for continuous feeding has reduced the stumpage loss, which was formerly as much as 50 per cent in some cases, and the electrodes are now much more uniform in character. Threaded amorphous carbon electrodes of foreign manufacture sell for about 2 to 3 cents per pound f. o. b. at the maker's plant. Domestic electrodes sell at about the same price but are not made threaded for continuous feeding. The use of graphite electrodes for steel furnaces is not economical because of the high cost per pound and because the consumption per ton of steel produced is almost as large as that of amorphous-carbon electrodes. The manufacturers of graphite electrodes claim that the gain in electrical conductivity offsets the higher cost, although the losses by heat conductivity would be greater unless the graphite electrodes were carefully designed.

For melting and refining cold scrap the consumption of amorphous-carbon electrodes varies from 15 to 50 pounds per ton. The consumption seems to be considerably higher in furnaces of the nonconducting-hearth type with two electrodes than in the conducting-hearth type with one electrode. The difference is probably due to the additional electrode, for in conducting-hearth furnaces of larger size having two to four electrodes the consumption per ton is greater than in the single-electrode furnace. In furnaces used for refining molten steel the wear on the electrodes is not as great and the consumption is much lower, being 6 to 12 pounds per ton.

THE LINING.

The rate of wear of the lining varies so much that it is almost impossible to state general figures. The furnace bottom will last 200 to 400 heats on cold charges and indefinitely on hot metal. The sides are repaired about every 100 heats but are not completely torn out. In most furnaces the hearth has a dolomite bottom and magnesite brick on the sides. The roof is generally silica brick, and, in an arc furnace, lasts 40 to 120 heats. In the arc furnaces operating under the best conditions the roof lasts about 70 heats. In furnaces with magnesite hearth, sides, and roof the lining cost per ton of metal is high. In some of the more recent furnaces the lining on the sides is about half as thick as that formerly used. The thinner lining increases the capacity and does not seem to affect the conduction losses enough to increase decidedly the power cost per ton of output. The hearth lining of an induction furnace lasts about 55 heats and the roof an indefinite period.

ESSENTIAL FEATURES OF REFINING IN AN ELECTRIC FURNACE.

The general process of refining steel in the electric furnace, as has been stated, comprises oxidation, followed by reduction or deoxidation. Owing to the nature of the heat supply, it is possible to have the atmosphere oxidizing, reducing, or neutral. This is not possible in the open-hearth furnace owing to the gases introduced for heating.

The function of slags[a] in refining steel in electric furnaces is somewhat different from that in refining in open-hearth furnaces. First, all oxygen introduced into the electric furnace must be in the solid form, whereas the open hearth has an unlimited supply in the air admitted. Oxidation can be conducted in an ordinary open-hearth furnace as in an electric furnace, except that the higher temperature of the latter hastens the melt. Deoxidation in which carbon is the chief agent can be performed only in the electric furnace.

During the oxidizing period sulphur is removed only to a small degree in the open-hearth furnace but much more efficiently in the electric furnace, especially when manganese ore is used. It is likely that part of the sulphur forms sulphur dioxide and passes off with the gases. The phosphorus is eliminated in the oxidizing period, just as in the open-hearth furnace.

Sulphur is removed in four ways: (1) As calcium sulphide, formed by the action of lime and carbon on ferrous sulphide at the high temperature of the arc furnace; (2) as calcium sulphide from the reaction of lime, ferrous sulphide, and calcium carbide at the higher temperatures of the arc furnace; (3) as calcium sulphide through the reaction of lime, ferrous sulphide, and silicon at the lower slag temperatures of the induction furnace; and (4) as iron sulphide from the action of ferrous oxide on calcium sulphide. The chemical reactions are as follows:

(1) $FeS+CaO+C=Fe+CaS+CO$.
(2) $3\ FeS+2\ CaO+CaC_2=3\ Fe+3\ CaS+2\ CO$.
(3) $2\ FeS+2\ CaO+Si=2\ Fe+2\ CaS+SiO_2$.
(4) $CaS+FeO=FeS+CaO$.

The final reaction is reversible, acting from left to right in an atmosphere even slightly oxidizing, whereas the action is from right to left in the first three reactions.

From the equations it may be seen that in both the induction and the arc furnace the formation of a highly basic slag is essential, but desulphurization in the former, owing to its lower temperature, is brought about chiefly by the reducing action of silicon upon the oxides, whereas in the latter the reducing agent is carbon at a higher temperature.

[a] Ambery, R., Slags in electric steel refining: Metall. Chem. Eng., vol. 10, 1912, p. 601.

In the arc furnace complete deoxidation is recognized by the presence of calcium carbide in the slag. When deoxidation of the bath is complete the contents of the electric furnace represent the nearest approach to ideal equilibrium between different chemical compounds that has ever been accomplished in large metallurgical operations. In the converter and the open hearth the metals are under the action of air and gas, in the crucible the metal takes up carbon and silicon, whereas in the electric furnace the action of the metal on the basic lining is very slight and there is no exchange of elements between metal and slag, except that if the dephosphorizing slag of the oxidizing period has not been completely removed before deoxidization begins some of the phosphorus in this slag and some in the new slag will be reduced. This last point must be carefully watched in all refining in the electric furnace.

Although the induction furnace can not attain the high temperature necessary for desulphurizing by use of calcium carbide as can arc furnaces, the results obtained in refining practice are about equal.

COST OF PRODUCING STEEL IN THE ELECTRIC FURNACE.

The cost of making steel in the electric furnace varies with local conditions. The cost of power does not enter so largely into the final cost as it does in some other electrometallurgical processes, especially the refining of molten steel. Plants are operating successfully under a power cost of 1 cent per kilowatt-hour in localities where material can be obtained at the price common to other processes. Plants such as the one at Ugine, France, have been established in remote localities, where the cost of power is very low, 0.2 cent per kilowatt-hour, but the cost of material is high.

COST OF PRODUCING STEEL IN THE GIROD FURNACE.

Girod[a] estimated in 1912 the cost of producing steel from cold scrap in the Girod furnace at Ugine as follows:

Cost of refining steel from cold scrap in Girod furnace at Ugine.

Item.	2.5 to 3 ton furnace.		10 to 12 ton furnace.	
Raw materials:				
Scrap, 1,100 kg., at $15 per ton	$16.50		$16.50	
Slag	.46		.46	
Deoxidizing and recarburizing additions	.70		.70	
		$17.66		$17.66
Producing cost:				
Electric power, 850 and 750 kilowatt-hours, at 0.2 cent per kilowatt-hour	3.40		3.00	
Electrodes, at $64 per ton	.60		.70	
Wages	.60		.30	
Maintenance and repairs	2.40		1.60	
		7.00		5.60
Total cost per ton of steel		24.66		23.26

[a] Girod, P., The electric steel furnace in foundry practice: Metall. Chem. Eng., vol 10, 1912, p. 668.

Girod estimated the cost of producing steel from molten steel at the same plant as follows:

Cost of refining steel from molten charge in Girod furnace at Ugine.

Item.	2.5 to 3 ton furnace.		10 to 12 ton furnace.	
Raw materials:				
Liquid steel, 4 per cent loss in heating, 1,040 kg., at $16 per ton	$16.64		$16.64	
Slags	.40		.40	
Deoxidizing additions	.70		.70	
		$17.75		$17.75
Producing cost:				
Electric power, 275 and 300 kilowatt-hours, at 0.2 cent per kilowatt-hour	1.10		.80	
Electrodes, at $64 per ton	.25		.25	
Wages, 8 heats per 24 hours	.20		.20	
Maintenance and repairs	.80		.50	
		2.35		1.75
Total cost per ton of steel		20.09		19.49

The estimates of Girod do not include expense for ingots, molds, superintendence, laboratory, amortization, general charges, or royalty charge. These figures apply only to conditions such as prevail at Ugine, where power is very cheap and the cost of material is high.

COST OF PRODUCING STEEL IN THE GRÖNWALL FURNACE, SHEFFIELD, ENGLAND.

The Electro-Metals Co., owners of the Grönwall patents, estimated the cost of production in 1912 for a 2-ton Grönwall furnace operating at Sheffield, England, as follows:

Cost of refining steel from molten charge, Grönwall furnace.

Item.	Without dephosphorizing.		With dephosphorizing.	
Number of charges per week	40	.....	30	.....
Tons per week	80	.....	60	.....
Cost per ton of steel:				
Power at 0.5 cent per kilowatt-hour	.....	$1.00	.....	$1.25
Repair of roof	.....	.16	.....	.20
Dolomite (100 pounds per charge)	.....	.17	.....	.17
Electrodes, at 4 cents per pound	.....	.18	.....	.22
Sundries	.....	.14	.....	.16
Royalty	.....	1.50	.....	1.50
	.....	3.15	.....	3.50

The items of labor, amortization, interest, raw materials, and general charges are omitted from the estimate given above. The figures refer to the production of steel with a sulphur and a phosphorus content below 0.02 per cent.

COST OF PRODUCING STEEL IN RÖCHLING-RODENHAUSER FURNACES.

Kjellin* in 1909 estimated the cost of production of high-grade steel for castings at Volklingen, Germany. The cost under German

* Kjellin, F. A., Induction and combination furnaces: Trans. Am. Electrochem. Soc., vol. 15, 1909, p. 172.

conditions with a 7-ton, three-phase Röchling-Rodenhauser furnace, using fluid basic Bessemer steel made at Volklingen, is estimated by Kjellin* as follows:

Cost of producing steel for castings in a 3-phase Röchling-Rodenhauser furnace, from molten metal.

	Cost per ton of steel.
Lining:	
Raw materials	$0.304
Wages	.043
Heating up furnace:	
2,000 kilowatt-hours at 2.17 cents for 160 tons	.272
Refining:	
300 kilowatt-hours per ton at 2.17 cents per kilowatt-hour	6.52
Wages	.425
Materials:	
Mill scale, 20 kg	.082
Lime, 30 kg	.087
Ferrosilicon, 6.5 kg	.485
Ferromanganese, 4 kg	.214
Power for cooling arrangements	.196
Repairs	.194
Total	8.822

To this must be added interest and the depreciation of the plant. Assuming 5 per cent interest and 10 per cent depreciation on $13,380, the approximate cost of the plant, gives $2,000 per year, which for 300 working days would give 41.2 cents per ton, so that the conversion of the fluid product from the basic converter into steel that can replace crucible steel for steel castings will cost $9.23 per ton exclusive of the cost of the molten steel charged. This cost is figured on a basis of three men operating the furnace.

Cost of producing rail steel in a 3-phase Röchling-Rodenhauser furnace, from fluid steel.

	Cost per ton of product.
Raw materials, steel at $15.54 per ton, and all fluxes	$16.56
Power at 0.58 cent per kilowatt-hour:	
Heating up	.02
Refining	.72
Wages:	
For furnace	.16
For lining	.008
Lining materials	.072
Tools	.097
Repairs	.194
Amortization and interest	.168
Licenses	.582
Total	18.62

This estimate includes all items. If the cost of molten steel be disregarded, the refining cost, with power at 0.58 cent per kilowatt-hour, will be $3.08 per ton.

* Loc. cit.

The cost of making casting steel of the composition given on page 105, from cold scrap in the Röchling-Rodenhauser furnace, at Lansdown, Pa., is estimated by the Crucible Steel Casting Co.,[a] as follows:

Cost of producing steel from cold scrap in the Röchling-Rodenhauser Furnace at Lansdowne, Pa.

	Cost per long ton of product.	
Scrap	$14.50	
Oxidation loss, about 2 per cent	.29	
		$14.79
Power, 844 kilowatt-hours at 0.7 cent per kilowatt-hour	5.91	
Fluxes	.76	
Ore	.05	
Labor	1.50	
Tools, repairs, and bottom	.76	
Air-cooling transformers	.12	
Conversion cost		9.10
Interest, 5 per cent; depreciation, 10 per cent per ton		1.50
Cost of 1 gross ton (2,240 pounds) electric steel ready to pour		25.39

PROPERTIES OF ELECTRIC-FURNACE STEEL.

For many years all high-grade steels were manufactured by the crucible process, but since the advent of the electric furnace there has been a gradual adoption of that furnace for refining steel. For the complete refining of the highest grades of steel the use of the electric furnace is now thoroughly established in Europe. Any product that can be made by the crucible process can be made by the electric furnace, and in most cases with cheaper raw materials and at a lower cost. In the electric furnace complex alloy steels can be made with precision. The high temperatures attainable facilitate the reactions and alloys need not be used so largely for the purpose of removing gas. Very low carbon steels can be kept fluid at the high temperatures. Steels free from impurities and of great value for electrical apparatus can be made. With the electric furnace large castings can be made from one furnace, whereas in the crucible process steel from several crucibles must be used. For small castings, which require a very high grade metal free from slags and oxides, electrically refined steel is especially adapted. The electric furnace gives a metal of low or high carbon content as desired, hot enough to pour into thin molds and still free from slags and gases.

There is now a tendency among consumers of rail and structural steel to require a higher-grade steel at an increased price rather than steel of acid Bessemer or even of basic open-hearth grade at a lower price. With the high cost of power that now prevails throughout the steel centers of the United States the electric furnace can not

[a] Von Baur, C. H., "The Röchling-Rodenhauser furnace of the Crucible Steel Casting Co., Lansdowne, Pa.: Metall. Chem. Eng., vol. 11, 1913, p. 113.

compete profitably with either the acid Bessemer or the basic open-hearth process in manufacturing steel of like grade from pig iron. It is in combination with either of these processes that the electric furnace seems destined to be prominent in steel manufacture. The cost of superrefining in the electric furnace the molten steel from either of these processes, exclusive of the cost of the molten steel, varies from $1.50 to $2.25 per ton, depending on the cost of power and the impurities to be removed.

Experiments conducted by the United States Steel Corporation[a] during the past four years show that, as compared with the acid Bessemer and basic open-hearth processes, the electric process has the following advantages: A more complete removal of oxygen; the absence of oxides caused by the addition of silicon, manganese, etc.; the production of steel ingots of 8 tons weight and smaller that are practically free from segregation; reduction of the sulphur content to 0.005 per cent if desired; reduction of the phosphorus content to 0.005 per cent, as in the basic open-hearth process, but with complete removal of oxygen.

Acid Bessemer steel has been refined in the basic electric furnace, yielding steel of the following composition: Carbon, 0.55 per cent; manganese, 0.137 per cent; silicon, 0.13 per cent; sulphur, 0.017 per cent; and phosphorus, 0.022 per cent. Rails made from this steel are comparatively soft, but show less wear than Bessemer rails in the same track and subjected to the same service. About 5,600 tons of standard electric steel rails from electric-furnace steel have been in service in the United States for the past two years. These rails have been subjected to all sorts of weather and to temperatures as low as —52° F. It seems that rails made by the basic electric process can be made softer than by either the acid Bessemer or basic open-hearth processes and yet show highly satisfactory wearing qualities.

No steel rails made by the basic electric process in service in this country have yet broken. Electric-furnace steel of a given tensile strength has a slightly greater elongation than basic open-hearth steel and is somewhat denser than basic open-hearth or acid Bessemer steel.

Below are given the averages of some comparative tests of rails from steel made in a Röchling-Rodenhauser furnace and Bessemer rails. The tests were made at Volklingen, Germany, for the Prussian State Railway.

[a] Clark, E. B., Various types and applications of electric steel furnaces: Metall. Chem. Eng., vol. 10, 1912, p. 378.

Comparison of steel rails made in Röchling-Rodenhauser furnace with basic Bessemer steel rails.

Kind of steel.	Carbon.	Silicon.	Manganese.	Phosphorus.	Sulphur.	Tensile strength per square inch.	Elongation.	Contraction.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Pounds.*	*Per cent.*	*Per cent.*
Electric	0.54	0.21	0.984	0.056	0.042	119,060	15.4	24.1
Bessemer	.36	.....	1.127	.073	.....	97,660	17.6	26.2

The results of some comparative tests, made at South Chicago, of electric-furnace steel for plates and basic open-hearth steel for plates were as follows:

Average ultimate strength and elongation of electric and open-hearth plate steels.[a]

Electric.			Open-hearth.		
Carbon content.	Ultimate strength per square inch.	Elongation on 2 inches.	Carbon content.	Ultimate strength per square inch.	Elongation on 2 inches.
Per cent.	*Pounds.*	*Per cent.*	*Per cent.*	*Pounds.*	*Per cent.*
0.08	59,194	27.25	0.08	51,690	32.00
.12	64,080	26.05	.12	56,510	29.70
.16	69,220	25.25	.16	52,901	28.61
.20	72,853	22.82	.20	58,294	28.82
.24	69,540	23.12	.24	63,560	26.25

[a] Osborne, G. C., The 15-ton Héroult furnace at the South Chicago works of the Illinois Steel Co.: Trans. Am. Electrochem. Soc., vol. 19, 1911, p. 221.

The results show a 15.5 per cent increase in ultimate strength and 11.3 per cent decrease in elongation for electric steel, as compared with open-hearth plate steel of approximately the same chemical composition.

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SUGGESTIONS FOR A DUPLEX PROCESS FOR MAKING STEEL.

Inasmuch as greater capacity and efficiency can probably be obtained by working an open-hearth furnace and an electric furnace in conjunction, the following description and the costs of working a 5-ton tilting basic open-hearth furnace and a 2½-ton electric furnace in this manner, as obtained from data furnished by Electro-Metals, Ltd., of London, England, are submitted here.

Although a 5-ton open-hearth furnace is rather small for keeping a 2½-ton electric furnace working at full capacity, yet the following method of procedure would give the maximum efficiency with the given plant, and serves to illustrate the point.

The open-hearth furnace should be charged at the beginning of the week, say at 12 p. m. Sunday. The charge under the special circumstances will consist of 4 tons of scrap and 1½ tons of pig iron, with the usual fluxes. This being the first charge, it will probably be ready to tap into the electric furnace at 6 a. m. Previous to this time the electric furnace has been heated with coke and with current and is ready to take a charge of hot metal.

At 6 a. m. 2½ tons of partly refined metal are drawn from the open-hearth furnace and put in the electric furnace. The open-hearth furnace is then recharged by putting pig and scrap on the banks and slopes of the furnace and allowing the cold material to come to almost a fusing temperature. Then, when the new material is hot enough it should be pushed into the bath and worked in the usual manner. The metal should be fit to tap in three to four hours.

Meanwhile it has taken about one hour to heat the crude fluid metal and to dephosphorize it, two hours more being taken to desulphurize the charge, so that by 9 a. m. it would be ready to tap, and by 9.15 the electric furnace would be ready to take another charge.

It is quite possible that the open-hearth furnace may not be ready for another hour or more. If this be the case, about 1 ton of cold scrap may be charged into the electric furnace and melted, by which time the open-hearth furnace will be ready to tap. This time it will be necessary to tap only 1½ tons of metal into the electric furnace. Owing to the small recharge in the open-hearth furnace the charge will be completely melted and perhaps partly refined by the time the electric furnace has finished its work. The electric furnace then being ready again at, say, 12.30, a 2½-ton charge of very hot refined metal will be tapped into it immediately, with no waste of time.

This method of assisting the open-hearth furnace can be carried out when necessary; perhaps at every alternate charge. One will see that the process presents these advantages:

The ton of scrap melted in the electric furnace will not need any pig iron to accompany it, as would be the case in the open-hearth furnace. That means that one-fourth ton of pig iron at $45 per ton is replaced by one-fourth ton of scrap at $5 per ton. Here is a clear saving of $10, which will pay for melting the ton of metal in the electric furnace many times over.

Although an "assisted" charge will take, say, 3½ hours to work, the resulting gain of time for the open-hearth furnace will reduce the time of the succeeding electric-furnace charge to less than 2½ hours and may even eliminate the dephosphorizing period in the electric furnace and thus reduce the time taken there to 2 hours.

The output by this method will be as follows:

One charge of 2½ tons could be run every 3½ hours, making 7 charges in 24 hours, or 42 casts per week of 6 days. This makes a weekly production of 105 tons, or a yearly production of 5,040 tons, allowing 48 weeks per year.

The cost of the process would be as follows:

Cost of producing steel by duplex process.

	Cost per ton.
Average power consumption, 420 kilowatt-hours, at 1.3 cents per kilowatt-hour	$5.46
Electrodes	.56
Slags	.62
Repairs to roof and lining	.66
Labor, 3 men each shift	.87
	8.17

It should be noted that there is a saving of $2 on every ton passing through the electric furnace by this method, owing to the use of scrap instead of pig iron.

DUPLEX PROCESS FOR LARGER FURNACE.

If the capacity of the open-hearth furnace be increased from 5 tons to, say, 7½ tons, then it may be found that the following method of increasing the output can be employed.

The open-hearth furnace should be charged with, say, 8 tons of scrap and pig iron. The phosphorus is then reduced to the proper limit, after which a 2½-ton charge can be transferred to the electric furnace for completion of the process. The electric furnace will be ready again in 2 hours for another charge.

It should be quite possible to recharge the open-hearth furnace and have the bath dephosphorized again by the time the charge in the electric furnace is finished. In this way a charge should be ready every 2 to 2½ hours.

The output will be as follows:

One charge of 2 tons could be run every 2 hours, making 12 charges in 24 hours, or 72 casts per week of 6 days. Therefore 130 to 140 tons would be produced in one week, or 6,200 to 6,700 tons in a year, allowing 48 working weeks to a year.

The cost of producing steel by this method would be as follows:

Cost of producing steel in duplex process, using 7½-ton furnace.

	Per ton of product.
Power consumption, 300 kilowatt-hours per ton, at 1.3 cent per kilowatt-hour	$3.90
Electrodes	.50
Slags	.37
Repairs to roof and lining	.56
Labor, 3 men each shift	.75
	6.08

COST OF OPERATING AN ELECTRIC FURNACE ALONE.

For comparison with the foregoing schemes, the following figures for making steel in the electric furnace alone are given:

In a 2½-ton electric furnace 1 cast can be made in 6 hours, or 4 casts in a day, making an output of 24 casts, or 60 tons per week. On a basis of 48 working weeks in a year, the yearly output will be 48 times 60, or 2,880 tons.

The cost of producing steel in the simple electric process would be as follows:

Cost of producing steel in 2½-ton electric furnace.

	Cost per ton.
Power, 950 kilowatt-hours per ton, at 1.3 cents per kilowatt-hour	$12.35
Electrodes	.87
Slags	.62
Repairs, roof, lining, and dolomite	.87
Labor, 3 men per shift	1.50
	16.21

Although the cost of operation is greater than that of the open-hearth, the cost per ton of the metal charged in the electric furnace is much less than that used in the open-hearth furnace, and this saving in cost should more than counterbalance the extra cost of working the electric furnace, not to mention the increased quality of the product.

A 5-ton electric furnace used for melting and refining would turn out 1 charge of 5 tons in 6 hours, or 4 casts in 24 hours. Twenty-four casts of 5 tons each, or 120 tons, would be produced in one week of six working days. The yearly production would be 5,760 tons, allowing 48 working weeks to a year.

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Bulletin .68

DEPARTMENT OF THE INTERIOR
BUREAU OF MINES
JOSEPH A. HOLMES, DIRECTOR

ELECTRIC SWITCHES

FOR USE IN GASEOUS MINES

BY

H. H. CLARK and R. W. CROCKER

WASHINGTON
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ELECTRIC SWITCHES FOR USE IN GASEOUS MINES.

By H. H. Clark and R. W. Crocker.

INTRODUCTION.

The purpose of the investigation discussed in this bulletin, one of a series dealing with the use of electricity in mines, was to study the various means and methods used to confine the flashes that occur when a switch carrying electric current is opened. The flash resulting from the opening of a 250-volt direct-current circuit carrying more than half an ampere will ignite explosive mixtures of mine gas and air. An equally dangerous flash may be produced by even less current from a 500-volt circuit. Few mine electric circuits carry less than half an ampere when in use. The need of protecting the switches of such circuits in places where gas is likely to be present is therefore obvious.

Two general methods have been proposed for preventing switching flashes from igniting gas surrounding the switch. One method is to inclose the switch in a casing provided with openings that are covered with wire gauze or otherwise designed and equipped for preventing the passage of flames from the interior to the exterior of the switch casing. Switches so protected are called explosion-proof switches. The second method is to immerse the switch contacts in oil to a depth sufficient to quench any flash that may occur when the switch is operated. Switches so protected are called oil switches.

The success of the first method of protection depends upon the proper design and construction of the switch casing. It is essential to the successful operation of the second method that the switch contacts be surrounded by the proper kind of oil in good condition. There are several ways in which an oil switch may be deprived of its protective feature. The oil tank may not be filled or, if filled, the oil may leak out or be spilled. The oil tank may be removed (in some designs) and not replaced or through neglect the condition of the oil may become such that it is no longer an efficient protection. While such contingencies are not likely to occur, and although oil switches may be so designed that the loss of oil is not probable, nevertheless, as compared with oil switches, explosion-proof switches seem to

possess a greater element of safety because their protective features are inherent in their construction, and for that reason are not likely, when needed, to be missing or impaired if the mechanical construction of the switches is sufficiently durable. Moreover, switches of the explosion-proof type can be so designed that the condition of their protective features may be readily observed each time that the switches are operated.

PRELIMINARY NATURE OF THE INVESTIGATION.

The investigation described in this bulletin was preliminary in nature and was undertaken principally to obtain information upon which to base future tests of a similar character. It was not the intention of the Bureau of Mines to approve any of the switches tested. The demand in this country for switches especially designed for mine service has not in the past been sufficiently great to interest manufacturers in the development of such switches. Consequently it is not surprising that only one of the switches submitted to the bureau for test appeared to be a practical design for general mine use.

CLASSIFICATION OF SWITCHES TESTED.

Mine switches may be divided into two classes according to their use. One class of switches may be connected permanently in circuit so that there is no chance for arcs to be formed by disconnecting the leads. The other class may be used under circumstances where it is necessary constantly to remove and replace the leads. In such cases the leads are not permanently connected and arcs may be formed by removing the leads while the switch is carrying load. This fact has led to the design of a type of switch in which the leads are so interlocked with the switch contacts that the leads can not be removed or replaced while the switch is closed.

The switches submitted for test were, for the purpose of this investigation, divided into two general types according to the construction of the switch and the character of the protective features. Each type was subdivided into two classes depending upon the manner in which the switch was designed to be connected into service, as follows:

Classification of the switches tested.

Type X, protected by explosion-proof devices.
  Class D, leads detachable.
  Class P, leads permanently connected.
Type O, protected by submerging the contacts in oil.
  Class D, leads detachable.
  Class P, leads permanently connected.

Eight switches were submitted for test. The table following shows the number and classification of the switches and gives a symbol by which each switch is designated in this report:

Number and designation of switches tested.

Number of switches submitted.	Type.	Class.	Designation symbol.
1	X	D	XD1
1	X	P	XP1
2	O	D	OD1 and OD2
4	O	P	OP1, OP2, OP3, and OP4

GENERAL CHARACTER OF TESTS.

The tests were originally intended to cover only type X switches which, in any size and capacity, could be tested without enlarging the generating equipment of the bureau. It seemed to be a foregone conclusion that type O switches would not ignite gas if the oil tank of the switch were properly filled with the proper kind of oil, unless the switches were opened under short-circuit conditions while connected directly across the terminals of a much larger generator[a] than was at the disposal of the bureau. Such a condition as last mentioned does not usually exist in the gaseous parts of coal mines and therefore oil switches would not have been included in this investigation had it not been that most manufacturers had nothing else to offer and earnestly requested that tests of oil switches be made. The available facilities of the bureau limited the capacity of the switches examined to 100 amperes and 225 volts.

METHOD OF TESTING TYPE X SWITCHES.

Type X switches were tested by operating them under load while the switch casings were filled and surrounded by explosive mixtures of gas and air. The flash caused by the operation of the switch invariably ignited the gaseous mixture contained within the switch casing, and the protective devices with which the switches were equipped were supposed to extinguish the resultant flames before they could issue from the casing of the switch and ignite the gas surrounding it.

METHOD OF TESTING TYPE O SWITCHES.

Type O switches were tested by operating them under load while the switch casings were filled and surrounded by explosive mixtures of gas and air. No ignition of gas took place as long as oil covered the switch contacts to the depth specified by the manufacturers.

[a] The capacity of the generator used in the tests was 200 kw. at 225 volts.

The facts determined by the test were therefore the extent to which the oil level of the various switches could be reduced by operation under load. The tendency of the oil to evaporate was determined by placing the switches in an inclosure where comparatively high temperatures were maintained for several weeks.

KIND OF GAS USED.

The natural gas supplied to the city of Pittsburgh was used in all tests. Its average composition at the time of the tests was approximately as follows: 83.1 per cent methane or marsh gas, 16.0 per cent ethane, 0.9 per cent nitrogen, and a trace of carbon dioxide. The term "gas" wherever used in this report refers to this natural gas. The most explosive mixture of such gas and air was determined to be one combined in the proportion of 8.6: 91.4.[a]

TESTING EQUIPMENT.

The equipment used in the tests included the following:

A gas-tight gallery or inclosure in which the switches were tested. This gallery was provided with a fan and accessory piping.

A device for determining the percentage of gas in the mixture.

A pressure indicator attached to the casings of type X switches to record the pressure developed therein by the explosion.

A rheostat for loading the switches with the proper current.

A system of piping and valves connecting the casings of type X switches with the fan intake for the purpose of filling the switch casings with the gaseous mixture that surrounded them.

Mechanisms for automatically operating the oil switches.

A small house in which the switches were mounted for the evaporation test.

TESTING GALLERY.

The gallery in which the switches were tested was a boiler-plate cylinder, 10 feet in diameter and 30 feet long, set horizontally upon a concrete foundation. Seven feet from each end the cylinder was stopped off with diaphragms of heavy manila paper reinforced with cheesecloth and painted with one coat of white lead and oil to make them gas tight. A manhole cut in the side of the gallery between the diaphragms made entrance possible without disturbing the paper stoppings. An exhaust fan, driven by an electric motor, was connected to the gallery from the outside for mixing and circulating the gas and air. The capacity of the fan, as connected, was about 1,000 cubic feet a minute. The gas-feed connection was made through a gas meter to the fan intake.

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[a] Tests by G. A. Burrell, chemist, Bureau of Mines.

GAS-PERCENTAGE INDICATOR.

A special instrument was developed to indicate directly the percentage of gas in the testing mixture. This instrument, which is described in detail in Bureau of Mines Bulletin 46,[a] showed, without delay, when the gaseous mixture passing through the fan contained the proper percentage of gas, and was especially helpful in determining whether or not the casings of type X switches had become filled with the gaseous mixture.

RECORDING PRESSURE INDICATOR.

An indicator of the type used in taking cards on gas engines was connected to the casings of the type X switches. The spring of the indicator was wound by turning the drum and was held in tension by a ratchet, which was released by an electromagnet at the same time that the igniting spark was applied. By this arrangement a curve was obtained that resembled half of an indicator diagram taken on an engine.

LOADING RHEOSTAT.

The loading rheostat consisted of cast-iron grids connected to a suitable controller, which was provided with means for automatic operation if desired.

GAS PIPES AND VALVES.

The casings of type X switches were connected to the intake of the fan by 1½-inch pipe provided with quick-acting valves that were held closed by a spring and opened by pulling a cord which was attached to the valve handle and was led to a point outside the gallery. Just before igniting the explosive mixture within the switch casing the observer closed these valves so that the only opening in the casings was through the protective devices.

MECHANISM FOR OPERATING THE OIL SWITCHES.

The mechanism for operating the oil switches consisted of a small, entirely inclosed motor, connected by gears to a system of levers that operated the nonautomatic type O switches in much the same way that they would be normally operated by hand. The number of operations was recorded automatically by a revolution counter attached to the operating mechanism.

The same mechanism, in connection with a cylinder and piston operated by air pressure, was used to connect the automatic type O switch to the power supply and to reset the switch after the rush of current had caused it to operate.

[a] Clark, H. H., An investigation of explosion-proof motors: Bull. 46, Bureau of Mines, 1913, p. 12.

EVAPORATION HOUSE.

The tests to determine the effect of evaporation upon the oil level of type O switches were conducted in a small house provided with electric heaters. The house was about 5 feet wide, 5 feet 6 inches long, and 7 feet 6 inches high inside. The air in this house was completely changed about three times an hour by natural ventilation.

SWITCH XD1.

DESCRIPTION.

Switch XD1 was a 200-ampere, 250-volt, double-pole, single-throw, slow-break, knife switch, mounted in a cast-iron casing composed of two parts hinged together. The details of construction are shown in Plate I, *A* and *B*. The switch blades consisted of National Electric Code inclosed fuses mounted on a swinging yoke. The switch was operated by a handle, *a*, pinned to a stud, *b*, that passed through the casing and constituted one of the pivots of the swinging yoke (Pl. I, *A*). Leads from the source of current entered the casing through gas-tight bushings, *j*, and were connected to the terminals, *i* (Pl. I, *B*). The connection to the load was made by inserting an attachment plug, *e*, into the receptacle, *c* (Pl. I, *B*).

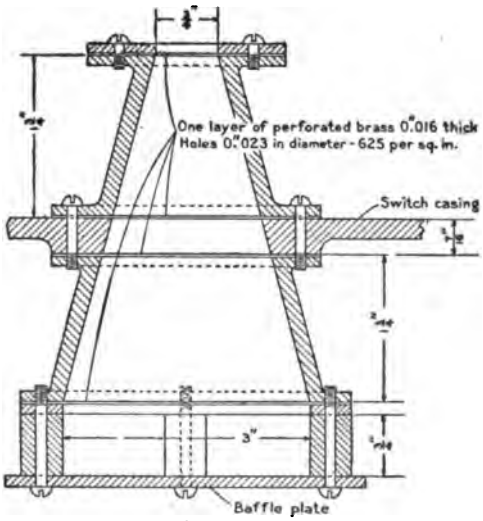
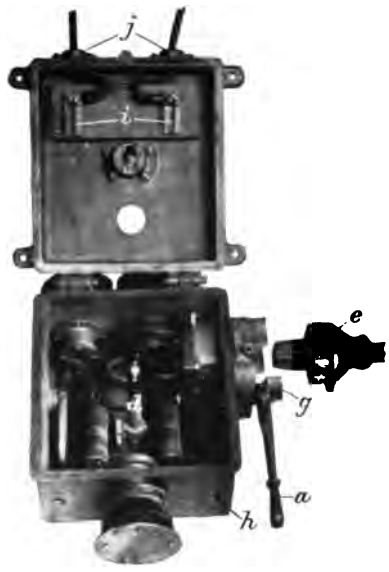


FIGURE 1.—Sectional view of switch XD1 protective devices.

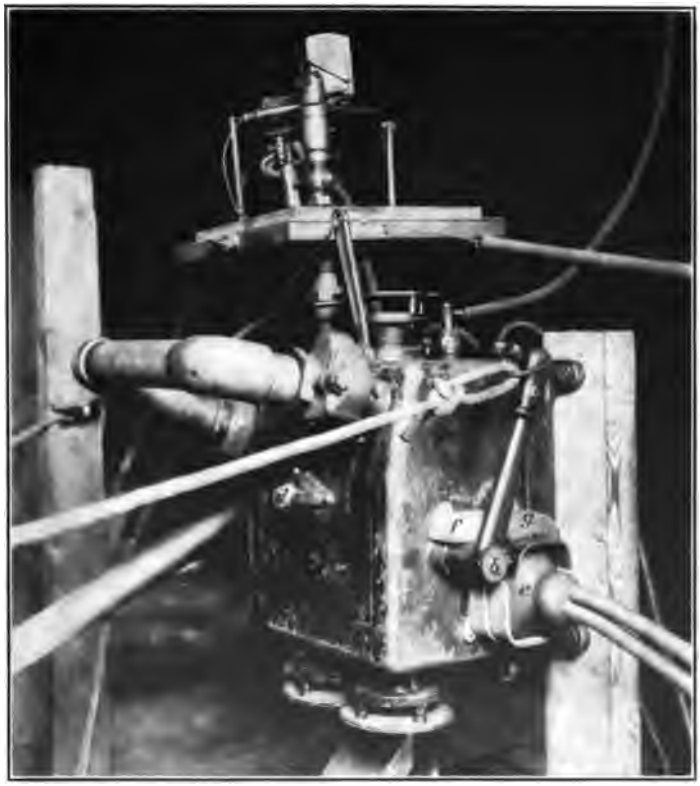
In order to prevent the improper operation of the switch, the switch blades, the attachment plug, and the two parts of the casing were interlocked so that it was impossible to manipulate the switch when the casing was open, to open the casing when the switch was closed, to close the switch before the attachment plug was in position, or to remove the attachment plug when the switch was closed.

The switch was equipped with three protective devices in the form of conical discharge tubes, or nozzles, each of which was provided with four disks of perforated metal, assembled as shown in figure 1. The devices were attached to the casing, as shown in Plate I, *A*, and furnished a total relief area of 0.356 square inch. The unoccupied space within the switch casing was estimated to be 0.46 cubic foot.

A. SWITCH XD1, FRONT VIEW, EXTERIOR.



B. SWITCH XD1, FRONT VIEW, INTERIOR.



C. SWITCH XD1, READY FOR TEST.

METHOD OF PROCEDURE.

The switch was set up in the testing gallery and connections of 1½-inch pipe were made from two opposite points in the switch casing (Pl. I, *C*) to the intake of the fan. Quick-acting valves were placed in the piping next to the casing so that it could be isolated from the fan piping before the gaseous mixture within the casing was ignited. With these valves open, the action of the fan created a partial vacuum in the casing, which was soon filled with gas drawn in through the openings of the protective devices.

The valves were so arranged that they could be operated from outside the gallery by means of a cord. The switch, in series with the loading rheostat, was connected to a source of direct current and a cord was provided for operating the switch from outside the gallery, so that the gaseous mixture within the switch casing might be ignited by the flash drawn when the switch was operated under load. Some of the tests were made in this manner, but in most of them a spark plug, *h* (Pl. I, *B*) was used for ignition purposes.

In making a test a card was placed on the drum of the recording pressure indicator, the quick-acting valves were opened, the testing gallery was closed, and the fan was started. The gallery was then filled with a properly proportioned mixture of gas and air. The gaseous mixture was then drawn through the switch casing by means of the fan until the gas percentage indicator showed that the casing was filled. All valves were then closed and the explosive mixture in the casing ignited. Observations of the phenomena accompanying the ignition were made and recorded. The maximum pressure developed in the casing by the explosion was determined by applying the proper scale to the curve on the indicator card. The gallery and switch casing were thoroughly aired before making another test.

RESULTS OF TEST.

Only a few tests were made on switch XD1 because the results obtained were so conclusive that nothing was to be gained by making more tests. Results of the tests are shown in the table following:

TABLE 1.—*Conditions and results of tests.*

Test No.	Character of ignition.[a]	Condition of protective devices.	Maximum pressure.	Puncture.[b]
			Pounds per square inch.	
1	S. P.	All open.	14.5	No.
2	Sw. 100	Do.	20.6	Yes.
3	S. P.	Do.	15.25	No.
4	S. P.	Do.	15.0	No.
5	S. P.	Do.	14.6	No.
6	S. P.	Do.	15.0	Yes.
7	Sw. 23.2	One lower plugged.	23.2	Yes.

[a] In this column "S. P." indicates spark-plug ignition. "Sw." indicates that the ignition was made by opening the switch under load, and the number following indicates the amperes of current flowing.

[b] For the sake of simplicity the term "puncture" has been applied to the ignition of the gaseous mixture surrounding the switch casing by flames discharged from the casing. In any tests where puncture is reported the protective devices failed to perform their safeguarding functions as evidenced by the fact that the gaseous mixture outside the casing was ignited by flames from within.

Flames were observed only at the protective device on the top of the switch, as the observer was not in a position to look into the outlets of the lower protective devices. However, a yellow light was observed to come from under the switch owing to the flames at the lower protective devices. The flames were yellow and did not extend beyond the edge of the baffle plate that protected the device, but extended about one-half or three-fourths of an inch beyond the outer plate of perforated metal. The first two punctures occurred at the lower protective devices. It was not possible to tell where the third puncture occurred.

DISCUSSION OF RESULTS OF TEST.

The results of the tests show that the switch is not suitable for use in gaseous atmospheres. Two punctures were obtained out of six trials made with the switch in its normal condition and another puncture was obtained in a trial made when one of the protective devices had been put out of commission. A puncture occurred in the only test made by igniting the mixture within the switch casing by a flash from the switch contacts when all the protective devices were in use. The protective devices failed to fulfill the purpose of their design, because there was not enough heat absorbing material in each partition of the protective devices and because these partitions were spaced too far apart.

REMARKS ON SWITCH DESIGN.

As regards mechanical strength, the design of the protective devices seemed to be satisfactory. The mechanical design of the devices for preventing the improper manipulation of the switch seemed to be good, although no tests were made to determine the wearing qualities of these devices.

SWITCH XP1.

DESCRIPTION OF SWITCH.

Switch XP1 was composed of two 600-volt, 40-ampere, single-pole, automatic circuit breakers inclosed in a cast-steel casing which was provided with protective devices. The details of construction are shown in Plate II, *A*, *B*. The circuit breakers were closed by the handles, *a* (Pl. II, *B*), which were connected through stuffing boxes in the cover to the forked arms *b*, operating the insulated switch levers, *c* (Pl. II, *A*). The automatic operation of the circuit breakers was accomplished by a solenoid of the usual type which, when excited by a predetermined current, released a latch that held the circuit

A. SWITCH XP1, INTERIOR VIEW.

B. SWITCH XP1, SET UP FOR TEST.

breakers closed. This latch could also be released from outside the casing by pressing upon the knobs, *d* (Pl. II, *B*), to which were connected rods which, passing through stuffing boxes in the cover, gave the latch a motion similar to that imparted by the solenoid.

The cover was secured to the casing by eight $\frac{3}{8}$-inch cap screws provided with collars to prevent their removal from the cover. Four locking bars, *g* (Pl. II, *A*), operated by the handle, *h*, were also provided and interlocked with the handles, *a* (Pl. II, *B*), so as to prevent not only the removal of the cover while the circuit breakers were closed, but also the closing of the circuit breakers while the cover was removed. Holes, *j* (Pl. II, *A*), were provided at the bottom of the switch casing for the entrance of leads. These holes were tapped for $\frac{3}{4}$-inch pipe, so that the wires could be run in conduits.

The unoccupied space within the switch casing was about 0.8 cubic foot.

The switch casing was equipped with four protective devices, *k*. These devices consisted of an outside plate of perforated brass about $\frac{1}{16}$ inch thick, immediately inside of which were two layers of 70-mesh copper gauze. Each plate of brass was perforated with 545 holes, approximately $\frac{1}{8}$ inch in diameter, spaced 0.2 inch apart. The total area of relief openings afforded by these devices was about 12.64 square inches.

METHOD OF PROCEDURE.

The procedure in the tests on switch XP1 was the same as that followed in the tests made on switch XD1 (page 11).

RESULTS OF TESTS.

Below are enumerated the conditions under which the tests were made, the maximum pressures developed by the explosion within the switch casing, the phenomena accompanying the explosion, and whether or not "puncture" (see Table 1, footnote [b], page 11) occurred.

Eighty-five tests were made on this switch under five different conditions. Eighty tests were made by ignition of the mixture from a flash at the contacts of the circuit breakers when they automatically opened. Five tests were made by ignition from a spark plug. Most of the tests were made with a gaseous mixture containing 8.6 per cent gas, which was believed to be the most explosive mixture. Some tests, however, were made with more and some with less than 8.6 per cent gas, in order to observe the effect of slow-burning mixtures. Some tests were made with coal dust sifted into the protective devices, in order to observe whether the dust particles would, while incandescent, be blown through the protective devices and ignite the gaseous mixture surrounding the switch casing.

In none of the tests were any flames seen to issue from the switch casing. In all tests the pressure developed within the switch casing by the explosion was comparatively low, due to the relatively large relief area of the protective devices.

In almost every test, immediately following the initial explosion, a burning of gas, accompanied by a loud humming noise lasting about 3 or 4 seconds, was seen at the inner face of the protective devices. The flames were blue, sometimes having a tinge of green toward the last, and appeared to move inward and upward. This is termed "afterburning." In those tests made with coal dust in which afterburning occurred, the flames appeared red, due to the incandescent dust particles. In all tests, except those made with coal dust, the flame of the initial explosion was reddish purple. When dust was used the initial flame was red.

In making the tests of group 2 (Table 2) it was found that the occurrence of afterburning depended on the quantity of dust that was sifted into the protective devices. When only a small quantity of dust was used afterburning occurred, but when a large quantity was used afterburning did not occur; also the presence of dust had a tendency to reduce the maximum pressure developed. In the tests made with 8.6 per cent gas alone, the average maximum pressure was 4.1 pounds; in the tests made with 8.6 per cent gas and a little dust, the average maximum pressure was 3.3 pounds; and in the tests made with 8.6 per cent gas and a large proportion of dust, the average maximum pressure was 2.7 pounds.

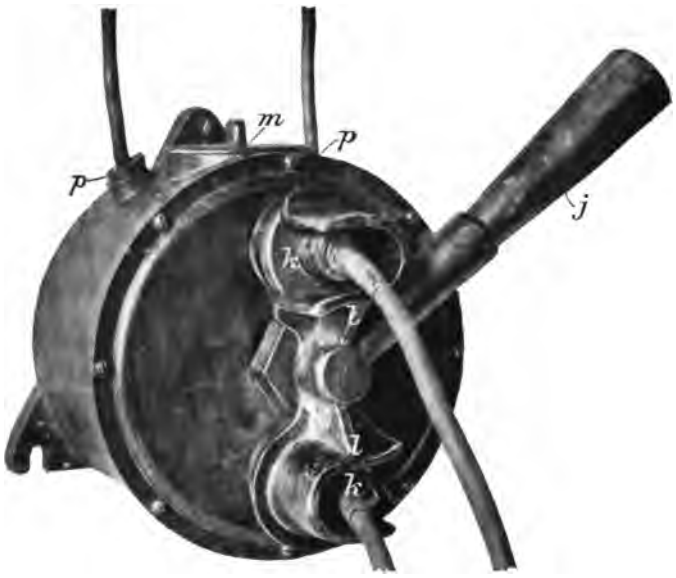
TABLE 2.—*Results of tests of switch XP1.*

Group No.	Gas.	Character of ignition.[a]	Maximum pressure.[b]	Number of tests made.	Number of punctures.[c]	Afterburning.
	Per cent.		*Pounds per square inch.*			
1	8.6	Sw.	4.12	35	0	Yes.
[d] 2	8.6	Sw.	3.18	15	0	8 Yes; 7 No.
3	8.6	Sp.	2.56	5	0	Yes.
4	7.0	Sw.	1.24	15	0	Yes.
5	10.0	Sw.	.67	15	0	Yes.

[a] In this column "Sp." indicates spark plug ignition. "Sw." indicates that the ignition was made by automatically opening the switch under load.
[b] The values given in this column are the average of all the maximum pressure observed in the tests of each group.
[c] See Table 1, footnote *b*.
[d] In this group of tests finely ground coal dust was sifted into the protective devices before each test.

The protective devices fulfilled the purpose of their design in that no puncture was obtained from 85 trials.[a]

[a] This statement, based upon the preliminary tests, does not mean that the switch is regarded as permissible for use in gaseous mines. Tests to establish a list of permissible electric switches are now being considered. (See section of this report headed "Permissible Switches for Use in Gaseous Mines," p. 30.)



A. SWITCH OD1, EXTERIOR VIEW.

B. SWITCH OD1, INTERIOR VIEW.

REMARKS ON SWITCH DESIGN.

The merits of the design are as follows: The ratio of relief opening area (12.64 square inches) to unoccupied space within the casing (0.8 cubic foot) is commendably high. Fine gauze (70-mesh) reinforced by perforated metal sheets appears to be an efficient device for absorbing heat.

The weak points in the design are as follows: The heat-absorbing material (perforated plates and gauzes) is not sufficiently protected from mechanical injury. The gauzes should be reinforced by a sheet of perforated metal on each side instead of on one side only. The unoccupied space within the switch casing appears to be greater than is necessary. The locking bars *g*, (Pl. II, *A*), alone did not draw the cover close enough to the casing to prevent the passage of flames. It was possible to open and close the circuit breakers while the cover of the switch casing was removed.

SWITCH OD1.

DESCRIPTION OF SWITCH.

This switch was a double-pole, single-throw, double-break oil switch, rated 100 amperes at 250 volts. The details of construction are shown in Plate III, *A* and *B*. The leads on the generator side of the switch passed through rubber-bushed holes, *p*, and connected with clips, *a*, mounted on fiber blocks. When the switch was closed, the blades, *c* and *d*, fitted into clips, *a* and *b*, respectively. The fuse holder, *e*, was provided with clips and these fitted onto knife blades, *f*. The various clips and blades were fastened in place and electrically connected by screws or rivets passing through the fiber insulators, *g*. All the moving parts were mounted on the shaft, *h*, which had a bearing at *i*, passed through a stuffing box in the cover, and had the handle, *j*, attached to its outer end. The load connection was made with attachment plugs, *k*, that fitted tightly into their receptacles, which were a part of the bases of clips, *b*. The wings, *l*, on the handle casting prevented the insertion or the removal of the plugs while the circuit was closed. The casing was of cast iron and was provided at the top with an opening, *m*, for putting in the oil and for renewing the fuses. This opening was closed by a screw cap. The fuses were mounted in such a position that they were accessible through this opening only when the switch was open. A plugged hole, *n*, provided a means for drawing off the oil without dismounting the switch. The cover was secured in place by screws and made tight by a lead gasket. The handles of the attachment plugs and the metal parts of the receptacles were protected by coverings of hard rubber. The grip of the handle was of hard wood. The casing could hold about 2.85 liters (approximately 3 quarts) of oil.

METHOD OF PROCEDURE AND RESULTS.

The switch was set up in the testing gallery and arranged so that it could be connected in series with the loading rheostat across the mains of a 200-kilowatt, 225-volt, direct-current generator. The rheostat was adjusted so that a current of 100 amperes flowed through the switch when it was closed. The switch was tested by opening and closing it by mechanical means about 8 times a minute. The test was continued in this manner until the switch had been opened and closed 28,980 times. During the progress of the test the switch contacts were renewed four times, but the oil was not changed. The life of each set of contacts was as follows:

Number of operations made with each set of contacts.

Contacts used.	Number of operations.
First set	2,453
Second set	4,939
Third set	10,878
Fourth set	4,489
Fifth set	6,221
Total	28,980

The temperature of the air that surrounded the switch varied from 10° C. in some tests to 24° C. in others. The average temperature of the oil throughout the tests was 33° C., the maximum being 87° C. when the contacts had become badly burned. The last 6,221 operations were made with the screw cap removed and the switch casing surrounded by the most explosive mixture of gas and air. The gas was not ignited in any test. Since no ignition occurred during the last 6,221 tests, it is probable that none would have occurred had gas been present during the 22,759 previous tests, and hence the test was equivalent to 28,980 operations in explosive gas.

The test was started with 3.25 liters (approximately 3 quarts) of oil [a] in the casing. Each time the contacts were changed some of the oil was unavoidably lost. Had this oil not been replaced it would have imposed an unfair condition upon the switch, and as it was almost impossible to determine the amount of oil lost during the process of renewing the contacts it was decided that after each such renewal the oil level should be brought up to its original height. This makes each series of runs, made with a single set of contacts, a test by itself, although in the latter runs the greater part of the oil had seen service on all previous runs. As many as 10,000 operations were made without lowering the oil level enough to expose the contacts, and as this may be taken to represent about three

[a] The analysis of this oil (No. 14-0) is given in Table 3.

years' service on a basis of 10 operations per day, it is reasonable to assume that the oil would be replenished before it had fallen to a dangerously low level through the operation of the switch.

During the first 22,000 operations about 250 c. c. (approximately one-half pint) of oil was forced out of the casing around the bushings provided for the incoming leads. No appreciable leakage occurred around the joint between the cover and the casing. The condition of the oil after the test had been made is shown in Table 4.

DISCUSSION OF RESULTS OF TEST.

The results of the tests indicate that as long as the switch contacts are submerged in the oil the switch, within its rated capacity, can safely be used in gaseous atmospheres.[a]

The absolute safety of the switch is, however, conditional upon the presence of oil in proper quantities. The tests show that the loss of oil due to evaporation and to the operation of the switch will not reduce the oil level to a dangerous point before the switch contacts have become so burned that the circuit can not be closed. The safety of the switch therefore depends largely upon the care that is taken to prevent the accidental loss of oil from the switch.

REMARKS ON SWITCH DESIGN.

The design of the switch is well suited to mine use. With one exception the mechanical strength of the parts seems to be sufficient to withstand the rough usage incident to underground service. The exception noted is the bushing provided for the incoming leads. It is suggested that the bosses on the casing be lengthened to allow a steel cap to be threaded onto each boss, and that the bushings be made of fiber and held between the caps and the face of the bosses. By this means the bushings would be held rigidly in place and protected from mechanical injury. It is further suggested that the switch contacts be placed so that each will be submerged in the oil to the same depth. This will have the effect of increasing the depth of oil over the contacts.

So far as was observed during the tests the joint between the cover plate and the casing allowed almost no leakage of oil. If the switches are not abused, the loss of oil from this joint should be very slight. The fuses were not tested, but it is recommended that inclosed fuses rather than open fuses be applied to this switch if such an arrangement is practicable.

[a] This statement, based upon the preliminary tests, does not mean that the switch is regarded as permissible for use in gaseous mines. See footnote [a], page 14.

SWITCH OD2.

DESCRIPTION OF SWITCH.

This switch was a double-pole, single-throw, 100-ampere, 600-volt, finger-contact, automatic oil switch. The details of construction are shown in Plate IV, *A* and *B*. The fingers *a* served to make the connection between the incoming terminals, *b*, and the load terminals, *c*, on the attachment plug, *d*. These fingers were mounted on a carriage that was, by the insertion of the plug, pushed backward against springs in chamber, *e*. Just after the plug entered its receptacle, *g*, the terminals, *c*, made connection with the contacts on the upper ends of the fingers, *a*, and at the same time the end of the plug came in contact with the end of the carriage upon which the fingers were mounted. By thrusting the plug farther into its receptacle the carriage was pushed backward until the fingers came in contact with the terminals, *b*, completing the circuit. The plug was held in position by a latch that engaged the tooth, *f*. When the switch was opened by hand, a slight upward pressure on the handle of the plug disengaged the latch, allowing the springs to force the carriage and plug outward, breaking the circuit between *a* and *b*, which were under oil. Thus the operation of the switch was accomplished by the insertion and removal of the plug. In the automatic opening of the switch the latch was disengaged by the release coil, *h*, and the operation completed as in the case of hand operation.

The attachment plug was of hard wood. The switch frame was of cast iron. The oil tank was of sheet metal with riveted and soldered seams, and could hold, up to the height designated by the manufacturer, about 4.7 liters (approximately 1¼ gallons) of oil.

METHOD OF PROCEDURE AND RESULTS.

The only test made upon this switch was a test to determine the evaporation of oil from the oil tank. The tank was filled with fresh oil and the switch placed in the evaporation house, where the temperature was maintained at 37° C. for 23 days and at 85° C. for 52 days of 24 hours each. The analyses of the oil (No. 15–0) before and after test are given in Table 3. This switch was examined after all the other oil switches had been tested. The previous tests demonstrated that as long as the contacts of oil switches of this capacity are covered to a reasonable depth with oil, flashes from the switch contacts can not ignite gas. These tests and the evaporation test made on this switch have also shown that the loss of oil due to evaporation and to the operation of the switch will not reduce the oil level to a dangerous point before the switch contacts have become so burned that the circuit can not be closed. In the light of the previous tests

A. SWITCH OD2, EXTERIOR VIEW.

SWITCH OD2, INTERIOR VIEW.

and from an examination of the switch and an analysis of the oil this oil switch as submitted should operate safely in gaseous atmospheres when carrying and breaking a current of 100 amperes from a 250-volt generator of at least 200-kilowatt capacity (the capacity of the generator available for the tests), and possibly from generators of greater capacity, especially if the switch were set up at some distance from the generator.[a]

The safety of the switch depends entirely upon the care that is taken to insure the presence of enough oil. The presence or absence of oil can not be seen by a glance at the exterior of the switch and can be determined only by removing the oil tank.

The safety of the switch is therefore conditional upon proper inspection, which can not be assured by test, and hence the absolute safety of the switch can not be vouched for.

REMARKS ON SWITCH DESIGN.

The design seems sufficiently simple and some of the parts are no doubt mechanically strong enough as submitted. The mechanism for preventing flashes from the contacts of the removable plug seems to perform its function. It is probable that the manufacturers contemplate further development of the switch before placing it upon the market. The removable plug seems to be the weakest and least developed part of the equipment, and must be materially improved in design and construction before it will be able to withstand satisfactorily the severities of underground service.

SWITCH OP1.

DESCRIPTION OF SWITCH.

This switch was a two-pole, 250-volt, 50-ampere, knife-blade, nonautomatic oil switch. The details of construction are shown in Plate V, *A*. The current-carrying parts were mounted on a porcelain base, *b*, approximately 5 by 5½ by 1½ inches, with the blades, *a*, and clips, *c*, on the under side, so that the arc was formed under about 4 inches of oil when the oil tank was filled to the height designated by the manufacturer. A wooden rod, *d*, for operating the blades passed through a hole in the center of the porcelain base. The leads entered the switch through porcelain bushings, *e*, at the top, and connected with connection sleeves on the upper side of the base.

The oil tank was made of sheet iron with welded seams and was secured to the switch frame by two bolts. The tank held about 2.9 liters (approximately 3 quarts) of oil[b] when filled to the height designated by the manufacturers.

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[a] This statement, based upon the preliminary tests, does not mean that the switch is regarded as permissible for use in gaseous mines. See footnote [a] on page 14.

[b] The analysis of the oil (No. 11-0) used in this switch is given in Table 3.

METHOD OF PROCEDURE AND RESULTS.

The switch was set up in the testing gallery and arranged so that it could be connected in series with the loading rheostat across the mains of a 200-kilowatt, 225-volt, direct-current generator. The rheostat was provided with an automatic device that allowed only 20 amperes to flow when the switch was first closed, but raised the current to 100 amperes just before the switch was opened. A tube was inserted through the side of the oil tank so that a thermometer could be suspended near the arc points of the switch contacts.

The switch was tested by opening and closing it by mechanical means about 11 times a minute. The test was continued in this manner until the switch had been operated 75,000 times. At frequent intervals explosive mixtures of gas and air were introduced to the testing gallery. No ignition occurred, as the oil level did not fall low enough to expose the contacts. At the end of the test the contacts were still submerged in oil to a depth of about 3¼ inches. The temperature of the air that surrounded the switch varied from −7° C. in some tests to 38° C. in others. The average temperature of the oil near the contacts was 17° C., the maximum being 77° C. when the contacts had become badly burned. During the progress of the test the switch contacts were renewed four times, but the oil was not changed. The following table shows the physical and chemical properties of the oils used in testing oil switches:

A. SWITCH OP1, INTERIOR VIEW.

B. SWITCH OP2, INTERIOR VIEW.

C. SWITCH OP3, INTERIOR VIEW.

TABLE 3.—*Analyses of oils [a] used with switches tested.*

Designating No. of oil used.	Switch in which oil was used.	Physical and chemical properties of the switch oils.											
		Specific gravity at 15° C.	Viscosity (°Engler at 20° C.).	Water.	Sulphur.	Acidity or alkalinity.	Fractional distillation.						
							Pressure (mm. of mercury).	Fractions distilled at increasing temperatures.				Residue.	Loss.
								225–250° C.	250–275° C.	275–300° C.	300–325° C.		
				Per cent.	*Per cent.*			*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
11–0	OP1	0.8760	8.0	0	0.04	Neutral	10	24.2	28.7	13.9	9.9	23.1	0.2
11–0	OP2	.8760	8.0	0	.04	do	10	24.2	28.7	13.9	9.9	23.1	.2
12–0	[b] OP3	.8575	4.6	0	Trace.	do	20	11.8	47.1	24.5	13.4	3.1	.1
12–0	[c] OP3	.8575	4.6	0	Trace.	do	20	11.8	47.1	24.5	13.4	3.1	.1
13–0	OP4	.9290	18.0	0	.19	do	14	26.2	31.9	24.1	15.2	2.3	.3
14–0	OD1	.9020	416.3	Trace.	.06	do	12	0	0	0	0	100.0	0
15–0	OD2	[d] .8725	9.1	0	.04	do	14	0	35.0	35.3	23.9	5.7	.1

Designating No. of oil used.	Physical and chemical properties of the switch oils.								Number of operations made.	Number of gallery ignitions.
	Condition at 0° C.	Free carbon.		Flash point. [e]		Ignition point. [f]	Loss during test.			
		Before test.	After test.	Before test.	After test.		Operation test.	Evaporation test.		
		Per cent.	*Per cent.*	*°C.*	*°C.*	*°C.*	*Per cent.*	*Per cent.*		
11–0	Liquid	0	4.45	158	135	202		2.2	75,000	0
11–0	do	0	3.95	158	132	202		4.0	75,000	0
12–0	do	0	4.19	170	105	195		3.9	3,471	1
12–0	do	0	3.89	170	145	195			2,775	1
13–0	do	0	4.35	166	85	187	9.8		75,000	0
14–0	Solid	Trace.	1.06	255	120	325	14.7 (?)		28,980	0
15–0	Liquid	0	.27	185	187	215		4.3	0	

[a] Analyses made by W. A. Jacobs, Bureau of Mines chemist.
[b] First series.
[c] Second series.
[d] Gravity after evaporation test was 0.8745.
[e] Pensky-Martens closed tester.
[f] Pensky-Martens open tester.

TABLE 4.—*Carbonization of No. 11–0 oil during the progress of the tests in which the oil was used.*

Designating No. of oil.	Switch in which the oil was used.	Per cent free carbon.													
		After 5,000 operations.	After 10,000 operations.	After 15,000 operations.	After 20,000 operations.	After 25,000 operations.	After 30,000 operations.	After 35,000 operations.	After 40,000 operations.	After 45,000 operations.	After 50,000 operations.	After 55,000 operations.	After 60,000 operations.	After 65,000 operations.	After 75,000 operations.
11–0	OP1	.....	.....	1.40	[a] 1.30	1.46	1.62	1.95	2.73	[a] 2.49	2.91	2.95	[a] 4.94	3.82	4.45
11–0	OP2	0.20	0.53	.....	1.19	1.27	.....	1.70	1.84	2.06	2.29	2.86	2.98	3.06	3.95

[a] The inconsistency of these results is probably due to the fact that a homogeneous sample of the oil was not obtained.

After each 5,000 operations a 26-gram (approximately 1 ounce) sample of oil was removed from the oil tank to determine the percentage of free carbon, 26 grams of fresh oil being added in each case. The results are shown in Table 4.

The life of each set of contacts was as follows:

Number of operations made with each set of contacts.

Contacts used.	Number of operations.
First set	[a] 13, 135
Second set	32, 503
Third set	9, 942
Fourth set	11, 680
Fifth set	[b] 7, 740
Total	75, 000

DISCUSSION OF RESULTS OF TEST.

The results of the tests indicate that as long as the switch contacts are submerged in oil to the depth recommended by the manufacturers the switch, within its rated capacity, can safely be used in gaseous atmospheres.[c]

The absolute safety of the switch is, however, conditional upon the presence of oil in proper quantities. The tests show that the loss of oil due to evaporation and to the operation of the switch will not reduce the oil level to a dangerous point before the switch contacts have become so burned that the circuit can not be closed. The safety of the switch therefore depends largely upon the care that is taken to prevent the accidental loss of oil from the switch.

REMARKS ON SWITCH DESIGN.

This switch was not designed especially for mine service and is not well adapted for rough usage. If mounted upon a switchboard in a well-constructed underground substation the switch should prove as satisfactory as if set up above ground, but for general use here and there about a mine under haphazard conditions of installation, such as mounting upon mine timbers or roughly constructed supports, the construction ought to be more substantial and the casing nearly airtight in order to preclude the loss of oil.

[a] Plate V, *A*, shows the condition of this set after the 13,135 operations had been made.

[b] This set was not burned off at the completion of the test.

[c] This statement, based upon the preliminary tests, does not mean that the switch is regarded as permissible for use in gaseous mines. See footnote [a], page 14.

SWITCH OP2.

DESCRIPTION OF SWITCH.

Switch OP2 was a two-pole, 6,600-volt,[a] 100-ampere, knife-blade, nonautomatic oil switch.

The details of construction are shown in Plate V, *B*. The switch was similar to switch OP2, except that the base was of hard wood. A wooden barrier about one-half inch thick separated the blades and clips of opposite polarity, thus dividing the oil tank into two parts. The oil tank was similar in construction to that of switch OP1, the inside dimensions were 7 by 7 by 9 inches, and the tank was lined throughout with fiber about $\frac{1}{16}$ inch in thickness. The tank held about 4.65 liters (approximately 5 quarts) of oil[b] when filled to the height designated by the manufacturers.

METHOD OF PROCEDURE AND RESULTS.

This switch was tested in exactly the same way as was the switch OP1. No ignition of the gaseous mixture occurred, as the oil level did not fall sufficiently to expose the contacts. At the end of the test the contacts were still submerged in oil to a depth of about 3¼ inches. The temperature of the air that surrounded the switch varied from −1° C. in some tests to 38° C. in others. The average temperature of the oil near the contacts was 31° C., the maximum being 110° C. when the contacts had become badly burned. During the progress of the test the switch contacts were renewed twice, but the oil was not changed. The life of each set of contacts was as follows:

Number of operations made with each set of contacts.

Contacts used.	Number of operations.
First set	[c] 13,650
Second set	51,350
Third set	[d] 10,000
Total	75,000

After each 5,000 operations a 26-gram (about 1 ounce) sample of oil was removed from the oil tank to determine the percentage of free carbon, 26 grams of fresh oil being added in each case. The results are shown in Table 4.

DISCUSSION OF RESULTS OF TEST.

The remarks on the results of the test of switch OP1, page 23, apply equally well to switch OP2.

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[a] Although this was the standard rating of the switch it was offered for test as a 250-volt switch.

[b] The analysis of the oil (No. 11-0) used in this switch is given in Table 3.

[c] Plate V, *B*, shows the condition of this set after the 13,650 operations had been made.

[d] These contacts were still in good condition when the test was completed.

REMARKS ON SWITCH DESIGN.

The remarks on the design of switch OP1, page 23, apply equally well to the design of switch OP2.

SWITCH OP3.

DESCRIPTION OF SWITCH.

This switch was a two-pole, 600-volt,[a] 100-ampere oil switch provided with an automatic overload release. The details of construction are shown in Plate V, *C*. The current-carrying parts were mounted on the under side of a wooden base *b*, approximately 6¾ by 7½ by ¾ inches. When the oil tank was filled to the designated level the contacts were submerged to a depth of about 2 inches.

The contact was a butt contact between two brass cylinders, *a* and *c*, about ¾ inch in diameter and 1 inch in length. The stationary contacts, *a*, were fastened directly against the lower side of the base by studs that passed through the base and terminated on its upper side in connection sleeves. The movable contacts, *c*, were supported on a wooden bar, *d*, and held in place against a spring (concealed in the bar) by studs that passed through the bar and connected through flat woven-wire braids, *f*, with the sleeves, *e*, to which were fastened the conductors entering the switch casing. The bar, *d*, was operated by two rods that passed through holes in the base and were attached to an operating lever. On one pole of the switch between the connection sleeve and the movable contact was located a solenoid, *g*, designed to open the switch on overload. The armature of this solenoid could be adjusted to act at any value of current between 80 and 160 amperes.

The frame of the switch was made of cast iron and was provided with six porcelain-bushed holes for the entrance of leads.[b]

The oil tank was made of sheet iron with welded seams. It measured 7 by 7½ by 8 inches and held about 5.25 liters (approximately 5½ quarts) of oil[c] when filled to the height designated by the manufacturers.

METHOD OF PROCEDURE AND RESULTS.

The switch was set up in the testing gallery and arranged so that it could be connected directly across the mains of a 200-kilowatt, 225-volt, direct-current generator. The resistance of the mains connecting the generator and the switch was measured and found to be about 0.37 ohm. The switch was tested by closing it and then

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[a] Although this was the standard rating of the switch it was offered for test as a 250-volt switch.

[b] The switch was evidently designed originally for three-phase service and had been only partly remodeled before submitting it for test.

[c] The analysis of the oil (No. 12-0) used in this switch is given in Table 3.

connecting it across the mains of the generator. After the rush of current had caused the switch to open automatically the connection between the switch and the generator was broken, the switch was once more closed, and the connection between switch and generator was again completed. The switch was manipulated by mechanical means at a rate of about 12 times a minute. An oil gage was attached to the oil tank in order that the change in height of the oil might be observed as the test progressed. By means of this gage the depth of oil in the tank could be determined with an accuracy of 1 per cent.

Arrangements were made for forcing an explosive mixture of gas into that part of the casing of the switch not occupied by oil. The testing gallery was not kept full of gas throughout all the tests because the explosive mixture would soon have been vitiated by the finely divided carbon that was given off in large quantities in a vapor-like form during the operation of the switch. It was considered best to operate the switch for a time and then to remove the carbon-charged air from the gallery and fill it with explosive gas, after which the operation of the switch was continued for another period before once more clearing the gallery and introducing more gas.

Two series of tests were made upon this switch. In the first series the automatic release was set to act at 160 amperes. The oil tank was filled with 5.2 liters (approximately 5½ quarts) of oil. The following table gives the rate of oil loss as the test proceeded and the number of times that gas was introduced into the gallery during the test:

TABLE 5.—*Rate of oil loss and number of operations made in gas while operating switch OPS, first series.*

Number of operations.	Loss of oil.	Number of operations made in gas.	Ignition of gas.
	C. c.		
0– 100	300	0	....
101– 520	350	0	....
521– 960	50	0	....
961–1,455	50	0	....
1,456–2,073	100	2,044–2,073	No.
2,074–2,560	70	0	....
2,561–3,470	30	2,561–2,600	No.
3,471	....	3,471	Yes.
	950		

After the gas had been ignited the switch was examined. It was found that the arc was formed just at the surface of the oil. As the ignition occurred on the first break after the introduction of the gas, and as nearly 900 breaks had been made just previously without gas in the gallery, the test was repeated in order to obtain more accurate data in regard to the number of times that the switch could be operated without igniting the gas.

The second test of this switch was made under the same conditions as the first. The following table shows the rate of oil loss and the number of times that gas was introduced to the gallery during the test:

TABLE 6.—*Rate of oil loss and number of operations made in gas while operating switch OP3, second series.*

Number of operations.	Loss of oil.	Number of operations made in gas.	Ignition of gas.
	C. c.		
0- 300	630	0	....
301- 590	70	0	....
591-1,270	70	0	....
1,271-2,260	110	0	....
2,261-2,773	40	2,261-2,280	No.
		2,664-2,683	No.
2,774-2,775	....	2,774-2,775	Yes.
	920		

After the switch had been operated 2,773 times gas was introduced in the gallery. The 2,774th operation did not cause the gas to ignite but the 2,775th operation did. An examination of the switch showed that the arc was formed under about $\frac{1}{16}$ inch of oil.

Fourteen times during the progress of the tests the switch contacts became so burned that they failed to complete the circuit, and it was necessary to adjust or renew them before the test could proceed. The number of operations made between each adjustment or renewal were as follows:

Life of contacts used in switch OP3 during the tests.

No. of test.	Number of operations.	Remarks.	No. of test.	Number of operations.	Remarks.
1	0	(a)	2	1,050	Adjusted.
1	520	Contacts adjusted.	2	1,270	Renewed.
1	1,455	Do.	2	1,552	Adjusted.
1	2,073	Renewed.	2	2,206	Do.
1	2,560	Adjusted.	2	2,260	Adjusted 1 pole, renewed 1 pole.
1	3,471	Test completed.			
2	0	Contacts renewed.	2	2,405	Contacts adjusted.
2	300	Adjusted.	2	2,663	Do.
2	590	Do.	2	2,775	Test completed.
2	640	Renewed.			

a The contacts that were on this switch when the test was started had been used for 650 operations previous to the beginning of this test.

DISCUSSION OF RESULTS OF TEST.

The remarks on the results of the tests of switch OP1 (p. 23) apply equally well to switch OP3.

REMARKS ON SWITCH DESIGN.

This switch was not especially designed for mine service. In fact it was not originally designed for direct-current service at all, but was a three-phase alternating-current switch adapted to direct-current service by removing one set of contacts and one of the overload release solenoids. The remarks on design of switch OP1 (p. 23) apply also to switch OP3.

SWITCH OP4.

DESCRIPTION OF SWITCH.

Switch OP4 was a three-pole, double-break, 250-volt, 100-ampere, nonautomatic oil switch of the manhole type. The details of construction are shown in Plate VI, *A*, *B*. The eccentric rod, *a*, opened and closed the switch by lowering and raising the wooden rods, *c*, which were attached to the contact brushes, *b* (Pl. VI, *B*). The eccentric rod was operated by the handle, *d*, on the outside of the switch casing (Pl. VI, *A*). The casing was made of cast iron about $\frac{5}{16}$ inch in thickness. The cover was provided with a lead gasket and secured by stud bolts. In setting up this switch in practice the lead sheath of the cable would be soldered to the brass sleeves, *e* (Pl. VI, A). The tank held about 7.5 liters (approximately 2 gallons) of oil[a] when filled to the height designated by the manufacturers.

METHOD OF PROCEDURE AND RESULTS.

The switch was installed in the testing gallery and arranged so that it could be connected, in series with the loading rheostat, across the mains of a 200-kilowatt, 225-volt, direct-current generator. The rheostat was adjusted so that 100 amperes flowed through the switch when it was closed.

The switch was tested by opening and closing it by mechanical means at a rate of about 10 operations per minute. The test was continued in this manner until the switch had been opened and closed 75,000 times. During the progress of the test the switch contact brushes were renewed three times, but the oil was not changed. The life of each set of contact brushes was as follows:

Number of operations made with each set of contacts.

Contacts used.	Number of operations.
First set	25,250
Second set	13,019
Third set	31,731
Fourth set[b]	5,000
Total	75,000

[a] The analysis of the oil (No. 13-0) used in this switch is given in Table 3.

[b] This set was still in good condition at the completion of the test.

A. SWITCH OP4, EXTERIOR VIEW.

B. SWITCH OP4, INTERIOR VIEW.

The temperature of the air that surrounded the switch varied from −3° C. in some tests to 22° C. in others. The average temperature of the oil throughout the tests was 35° C., the maximum being 91° C. when the contacts had become badly burned. The last 5,000 operations were made while the switch casing was open and surrounded by the most explosive mixture of gas and air. The gas was not ignited in any test.

Since no ignition occurred during the last 5,000 operations it is probable that none would have occurred had gas been present during the 70,000 previous operations, and hence the test was equivalent to 75,000 operations in explosive gas.

Table 3 shows that during the test the oil decreased 9.8 per cent in weight. This was due first to the escape of oil vapors through the openings designed to be completely filled by the leads entering the switch, but not so filled in this test; and second to the unavoidable loss of oil incident to the removal of the burned contact brushes and the substitution of new ones.

DISCUSSION OF RESULTS OF TEST.

The remarks on the results of test of switch OP1 (p. 23) apply also to switch OP4.

REMARKS ON SWITCH DESIGN.

The design and construction of the switch seems to be sufficiently substantial for mine service. The casing is so designed that leakage of oil is practically impossible. If the switch is to be used with any but lead-covered conductors the openings for the admission of such conductors to the casing will have to be of a form different from that used in the switch that was tested.

REMARKS ON THE GENERAL DESIGN OF AN ELECTRIC MINE SWITCH.

The purpose for which a switch is to be used will of course influence its design. In general, a mine switch is subjected to more severe service and to more abuse than a switch that is used above ground. Therefore a mine switch should be constructed more substantially than a switch for ordinary service. The voltage requirements of mine switches are not severe at present, although as the use of alternating current becomes more extensive higher voltages will no doubt be used underground. However, in places where there is danger of gas ignition a voltage of 440 for alternating current and 650 for direct current will probably never be exceeded. The current requirements are not unusual. The majority of mine switches are nonautomatic, fuses being used to open the circuits on overload.

Although often used merely as disconnecting switches, all switches for mine service should be capable of opening under at least twice their rated current load without blowing the fuse or damaging the connections.

There may be two classes of safety mine switches—those designed to be permanently connected into circuit and those so designed that the leads connecting the switch with its load are readily detachable. The latter class requires an interlocking connection between the detachable leads and the blades of the switch so that the circuit can not be opened by the removal of the detachable leads. The classification of safety mine switches has already been discussed in pages 6 and 7.

PERMISSIBLE SWITCHES FOR USE IN GASEOUS MINES.

As a result of the preliminary tests reported in the foregoing pages the bureau has decided to undertake tests to establish a list of permissible electric switches for use in gaseous mines.

The following is a statement of the character of the tests, the conditions under which they will be made, and the requirements for approval.

DEFINITIONS OF TERMS USED.

Permissible.—The Bureau of Mines considers a switch permissible if it is similar in all respects to the sample switch that passed certain tests made by the bureau, and if it is set up and used in accordance with the conditions prescribed by the bureau.

Permissible switches may be either of the oil type or of the explosion-proof type. Switches of either type may belong to the class having detachable leads (class D) or the class having permanently connected leads (class P).

Types.—The Bureau of Mines has applied the term "explosion-proof" to switches so constructed as to prevent the ignition of mine gas surrounding the switch by any sparks, flashes, or explosions of gas that may occur within the switch casing.

Oil-type switches are those that have their contacts immersed in oil for the purpose of confining the flash that takes place when the switch is opened under load.

REQUIREMENTS FOR APPROVAL.

IDENTIFICATION OF SWITCHES.

Before the Bureau of Mines approves any switch as permissible there shall be on file with the bureau the following data to be used for the identification of the switch and to be mentioned in the published approval of the switch:

1. The voltage and current rating of the switch.

2. Dimension drawings that show clearly—(*a*) the size and general appearance of the switch casing; (*b*) the relative arrangement of parts within the switch casing; (*c*) for explosion-proof switches the size and details of the protective devices and their relative arrangement on the switch casing; (*d*) for oil switches the trade name of the oil that should be used in connection with the switch; (*e*) any other drawings necessary to identify or explain any feature that is to be considered in the approval of the switch.

DESIGN AND CONSTRUCTION.

The design and construction of permissible mine switches must be especially substantial. This requirement shall be applied consistently to all the details of the switches as well as to their principal parts in order to give assurance that under the severe conditions of mining service the explosion-proof qualities of the switches will remain unimpaired.

The protective devices used with permissible explosion-proof switches must not only be capable of preventing the passage of flames from the interior to the exterior of the switch casing, but such devices must also possess sufficient mechanical strength to prevent the accidental impairment of their usefulness. If there are movable parts in connection with such devices, such parts must be so designed that there can be no interference with their movement. All joints in the casing of an explosion-proof switch must be metal-to-metal joints with broad faces. All openings in the casing of an explosion-proof switch other than those provided with protective devices by the manufacturers must be tightly closed. It is desirable that such openings be as few as possible. There must be no exposed terminals or contacts outside the switch casing. The leads connecting the switch with the power supply must enter the switch casing through gas-tight bushings of approved design and be provided with lugs for making permanent connections within such casings. If there are glass-covered openings in the casing of a switch, the glass must be protected by a metal cover or covers that close automatically unless held open. Class P explosion-proof switches must have the covers or doors of the switch casing so interlocked with the switch mechanism that the casing can not be opened when the switch is closed, nor can the switch be operated while the casing is open. Class D explosion-proof switches must be equipped like class P switches, and in addition have their detachable leads so interlocked with the switch mechanisn that the leads can not be removed while the switch is closed.

Oil-type switches must be so designed and constructed as to reduce to a minimum the possibility of oil leaking from the switch casings. A practically air-tight casing is recommended. The switch contacts

must be well submerged in oil. All leads must leave the casing at points above the designated oil level. Class D oil switches must have their leads interlocked with the switch mechanism in the same manner as is prescribed for Class D explosion-proof switches.

CHARACTER OF TESTS.

TESTS OF EXPLOSION-PROOF SWITCHES.

In testing an explosion-proof switch for permissibility the switch casing will be filled and surrounded with the most explosive mixture of Pittsburgh natural gas and air. The mixture within the casing will then be ignited by means of a spark plug, by a flash from the switch contacts, or by any other means that simulates the conditions of actual practice. Similar tests will also be made with greater and with less proportions of gas in the explosive mixture and with coal dust sifted into the switch casing or into the protective devices. Tests will also be made to determine the point of ignition that gives the greatest pressure, and tests will be made by igniting from such a point. Not less than 50 tests of all kinds shall be made and more than that number may be made if, in the opinion of engineers of the bureau, more tests are necessary to prove the permissibility of the switch. In order to pass these tests, a switch must in none of them cause an ignition of the gas surrounding the switch or the discharge of flames from any part of the switch casing.

A switch will not be regarded as permissible, although it may have passed the tests just described, if used (*a*) without the caution plate described on page 33 or (*b*) with openings in the switch casing other than those openings provided with protective devices by the manufacturer.

EXPLOSION-PROOF FUSE CASINGS.

Only inclosed fuses should be used in connection with explosion-proof or oil switches. The fuses should be inclosed in explosion-proof casings or immersed in oil. The former practice is recommended because it renders the fuses easier of access when replacements are necessary.

If fuses are inclosed in explosion-proof casings, such casings will be tested in the same manner as switch casings of similar design. If fuses are submerged in oil, the investigation of this part of the equipment will consist of a thorough examination of the details of design and construction.

TESTS OF OIL-TYPE SWITCHES.

In testing an oil-type switch for permissibility the switch will be set up in an inclosure filled with the most explosive mixture of natural gas and air. The contacts of the switch will be covered with

oil to only 25 per cent of the depth designated by the manufacturer. Under these conditions 50 tests will be made by opening the switch under rated conditions of current and voltage. Direct current will be used in all tests. In order that a switch may successfully pass these tests its operation shall in none of them ignite the gas surrounding the switch. The mechanical design and construction of the switch will be considered in its approval.

Even if a switch has passed the tests just described it will not be considered as permissible if used without the caution plate described below.

CAUTION AND APPROVAL PLATES.

As part of the protection of a permissible switch the manufacturers shall be required to attach to an explosion-proof switch a metal plate inscribed as follows:

> CAUTION.
>
> The permissibility of this switch depends upon the absence of any openings in the casing other than those provided with protective devices by the manufacturer.
>
> The casing should be frequently inspected for improper openings.

And to an oil-type switch a metal plate inscribed as follows:

> CAUTION.
>
> The permissibility of this switch depends upon the presence of the proper quantity of ———— oil in the switch casing.
>
> The oil level should be carefully maintained at the point designated by the manufacturer.

The manufacturer shall be permitted to attach to the casing of an explosion-proof switch a plate inscribed as follows:

PERMISSIBLE

EXPLOSION-PROOF SWITCH

For use with permanent connections (or for use with detachable leads).

Approval No. ———.

U. S. BUREAU OF MINES.

And to the casing of an oil-type switch a plate inscribed as follows:

PERMISSIBLE

OIL SWITCH

For use with permanent connections (or for use with detachable leads).

Approval No. ———.

U. S. BUREAU OF MINES.

The size, material, and design of both caution and approval plates shall meet with the approval of the bureau.

The caution and statement of approval could be combined upon a single plate, a suggested form for which is shown.

PERMISSIBLE
EXPLOSION-PROOF SWITCH
FOR USE WITH DETACHABLE LEADS

APPROVAL NO. _____
U. S. BUREAU OF MINES

CAUTION.

The permissibility of this switch depends upon the presence in the switch casing of the proper quantity of _____ oil.

The oil level should be carefully maintained at the point designated by the manufacturer.

NOTIFICATION OF MANUFACTURER.

As soon as the engineers of the bureau are satisfied that a switch is permissible for use in places where gas may occasionally be present in such proportions as to form an explosive mixture, the manufacturer of the switch shall be notified to that effect. As soon as a manufacturer is formally notified that his switch has passed the tests prescribed by the bureau he shall be free to advertise such switch as permissible and may attach approval plates to such switches.

SCOPE OF APPROVAL.

The approval of any switch by the bureau shall be construed as applying to all switches of that specific type, class, form, and rating made by the same manufacturer and protected in the same manner, but to no other switches.

Manufacturers shall, before claiming approval by the bureau for any modification of any approved switch, submit to the bureau drawings that shall show the extent and nature of such modifications in order that the bureau may decide whether or not it will be necessary to test the remodeled switch before approving it. Each approval of a permissible switch will be given a serial number. Approvals of modified forms of a previously approved switch will bear the same number as the original approval with the addition of the letters A, B, C, etc.

WITHDRAWAL OF APPROVAL.

The bureau reserves the right to rescind for cause at any time any approval granted under the conditions herein set forth.

PRECAUTIONS.

It is obviously futile to protect the switch and neglect to safeguard any spark-producing accessory apparatus. It is equally unavailing to protect all of the apparatus if, within the limits made dangerous by the presence of gas, there are used in connection with the electrical equipment uninsulated wires or wires not properly connected with suitable insulators.

PUBLICATIONS ON MINE ACCIDENTS AND METHODS OF MINING.

The following Bureau of Mines publications may be obtained free by applying to the Director Bureau of Mines, Washington, D. C.

BULLETIN 10. The use of permissible explosives, by J. J. Rutledge and Clarence Hall. 1912. 34 pp., 5 pls., 4 figs.

BULLETIN 17. A primer on explosives for coal miners, by C. E. Munroe and Clarence Hall. 61 pp., 10 pls., 12 figs. Reprint of United States Geological Survey Bulletin 423.

BULLETIN 20. The explosibility of coal dust, by G. S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz. 204 pp., 14 pls., 28 figs. Reprint of United States Geological Survey Bulletin 425.

BULLETIN 42. Apparatus for the analysis of mine gases, by G. A. Burrell. 1913. — pp., — pls.

BULLETIN 44. First national mine-safety demonstration, Pittsburgh, Pa., October 30 and 31, 1911, by H. M. Wilson and A. H. Fay, with a chapter on the explosion at the experimental mine, by G. S. Rice. 1912. 75 pp., 7 pls., 4 figs.

BULLETIN 45. Sand available for filling mine workings in the Northern Anthracite Coal Basin of Pennsylvania, by N. H. Darton. 1913. 33 pp., 8 pls., 5 figs.

BULLETIN 46. An investigation of explosion-proof mine motors, by H. H. Clark. 1912. 44 pp., 6 pls., 14 figs.

BULLETIN 48. The selection of explosives used in engineering and mining operations, by Clarence Hall and S. P. Howell. 1913. 50 pp., 3 pls., 7 figs.

BULLETIN 50. A laboratory study of the inflammability of coal dust, by J. C. W. Frazer, E. J. Hoffman, and L. A. Scholl, jr. 1913. 60 pp., 95 figs.

BULLETIN 51. The analysis of black powder and dynamite, by W. O. Snelling and C. G. Storm. 1913. 80 pp., 5 pls., 5 figs.

BULLETIN 52. Ignition of mine gases by the filaments of incandescent electric lamps, by H. H. Clark and L. C. Ilsley. 1913. 31 pp., 6 pls., 2 figs.

BULLETIN 53. Mining and treatment of feldspar and kaolin in the southern Appalachian region, by A. S. Watts. 1913. 171 pp., 16 pls., 12 figs.

BULLETIN 56. First series of coal-dust tests in the experimental mine near Bruceton, Pa., by G. S. Rice, L. M. Jones, J. K. Clement, and W. L. Egy. 1913. 115 pp., 12 pls., 28 figs.

BULLETIN 59. Investigations of detonators and electric detonators, by Clarence Hall and S. P Howell. 1913. 79 pp., 10 pls., 5 figs.

BULLETIN 62. Record of the national mine-rescue and first-aid conference, Pittsburgh, Pa., September 23, 25, 1912, by H. M. Wilson. 1913. 74 pp.

BULLETIN 65. Oil and gas wells through workable coal beds; papers and discussions, by G. S. Rice, O. P. Hood, and others. 1913. 101 pp., 1 pl., 11 figs.

TECHNICAL PAPER 6. The rate of burning of fuse as influenced by temperature and pressure, by W. O. Snelling and W. C. Cope. 1912. 28 pp.

TECHNICAL PAPER 7. Investigation of fuse and miners' squibs, by Clarence Hall and S. P. Howell. 1912. 19 pp.

TECHNICAL PAPER 11. The use of mice and birds for detecting carbon monoxide after mine fires and explosions, by G. A. Burrell. 1912. 15 pp.

TECHNICAL PAPER 13. Gas analysis as an aid in fighting mine fires, by G. A. Burrell and F. M. Seibert. 1912. 16 pp., 1 fig.

TECHNICAL PAPER 14. Apparatus for gas-analysis laboratories at coal mines, by G. A. Burrell and F. M. Seibert. 1913. 24 pp., 7 figs.

TECHNICAL PAPER 17. The effect of stemming on the efficiency of explosives, by W. O. Snelling and Clarence Hall. 1912. 20 pp., 11 figs.

TECHNICAL PAPER 18. Magazines and thaw houses for explosives, by Clarence Hall and S. P. Howell. 1912. 34 pp., 1 pl., 5 figs.

TECHNICAL PAPER 19. The factor of safety in mine electrical installations, by H. H. Clark. 1912. 14 pp.

TECHNICAL PAPER 21. The prevention of mine explosions; report and recommendations, by Victor Watteyne, Carl Meissner, and Arthur Desborough. 12 pp. Reprint of United States Geological Survey Bulletin 369.

TECHNICAL PAPER 22. Electrical symbols for mine maps, by H. H. Clark. 1912. 11 pp., 8 figs.

TECHNICAL PAPER 24. Mine fires, a preliminary study, by G. S. Rice. 1912. 51 pp., 1 fig.

TECHNICAL PAPER 28. Ignition of mine gas by standard incandescent lamps, by H. H. Clark. 1912. 6 pp.

TECHNICAL PAPER 29. Training with mine-rescue breathing apparatus, by J. W. Paul. 1912. 16 pp.

TECHNICAL PAPER 30. Mine-accident prevention at Lake Superior iron mines, by D. E. Woodbridge. 1913. 38 pp., 8 figs.

TECHNICAL PAPER 31. Apparatus for the exact analysis of flue gas, by G. A. Burrell and F. M. Seibert. 1913. 12 pp., 1 fig.

TECHNICAL PAPER 33. Sanitation at mining villages in the Birmingham district, Ala., by D. E. Woodbridge. 1913. 27 pp., 1 pl., 9 figs.

TECHNICAL PAPER 40. Metal-mine accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 54 pp.

TECHNICAL PAPER 44. Safety electric switches for mines, by H. H. Clark. 1913. 8 pp.

TECHNICAL PAPER 46. Quarry accidents in the United States during the calendar year 1911, compiled by A. H. Fay. 1913. 32 pp.

TECHNICAL PAPER 47. Portable electric mine lamps, by H. H. Clark. 1913. 13 pp.

TECHNICAL PAPER 48. Coal-mine accidents in the United States, 1896–1912, with monthly statistics for 1912, compiled by F. W. Horton. 1913. 74 pp., 10 figs.

TECHNICAL PAPER 52. Permissible explosives tested prior to March 1, 1913, by Clarence Hall. 1913. 11 pp.

MINERS' CIRCULAR 3. Coal-dust explosives, by G. S. Rice. 1911. 22 pp.

MINERS' CIRCULAR 4. The use and care of mine-rescue breathing apparatus, by J. W. Paul. 1911. 24 pp., 5 figs.

MINERS' CIRCULAR 5. Electrical accidents in mines; their causes and prevention, by H. H. Clark, W. D. Roberts, L. C. Ilsley, and H. F. Randolph. 1911. 10 pp., 3 pls.

MINERS' CIRCULAR 6. Permissible explosives tested prior to January 1, 1912, and precautions to be taken in their use, by Clarence Hall. 1912. 20 pp.

MINERS' CIRCULAR 7. The use and misuse of explosives in coal mining, by J. J. Rutledge. 1913. 52 pp., 8 figs.

MINER'S CIRCULAR 8. First-aid instructions for miners, by M. W. Glasgow, W. A. Raudenbush, and C. O. Roberts. 1913. 66 pp., 46 figs.

MINERS' CIRCULAR 9. Accidents from falls of roofs and coal, by G. S. Rice. 1912. 16 pp.

MINERS' CIRCULAR 10. Mine fires and how to fight them, by J. W. Paul. 1912. 14 pp.

MINERS' CIRCULAR 11. Accidents from mine cars and locomotives, by L. M. Jones. 1912. 16 pp.

MINER'S CIRCULAR 12. The use and care of miners' safety lamps, by J. W. Paul. 1913. 16 pp., 4 figs.

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